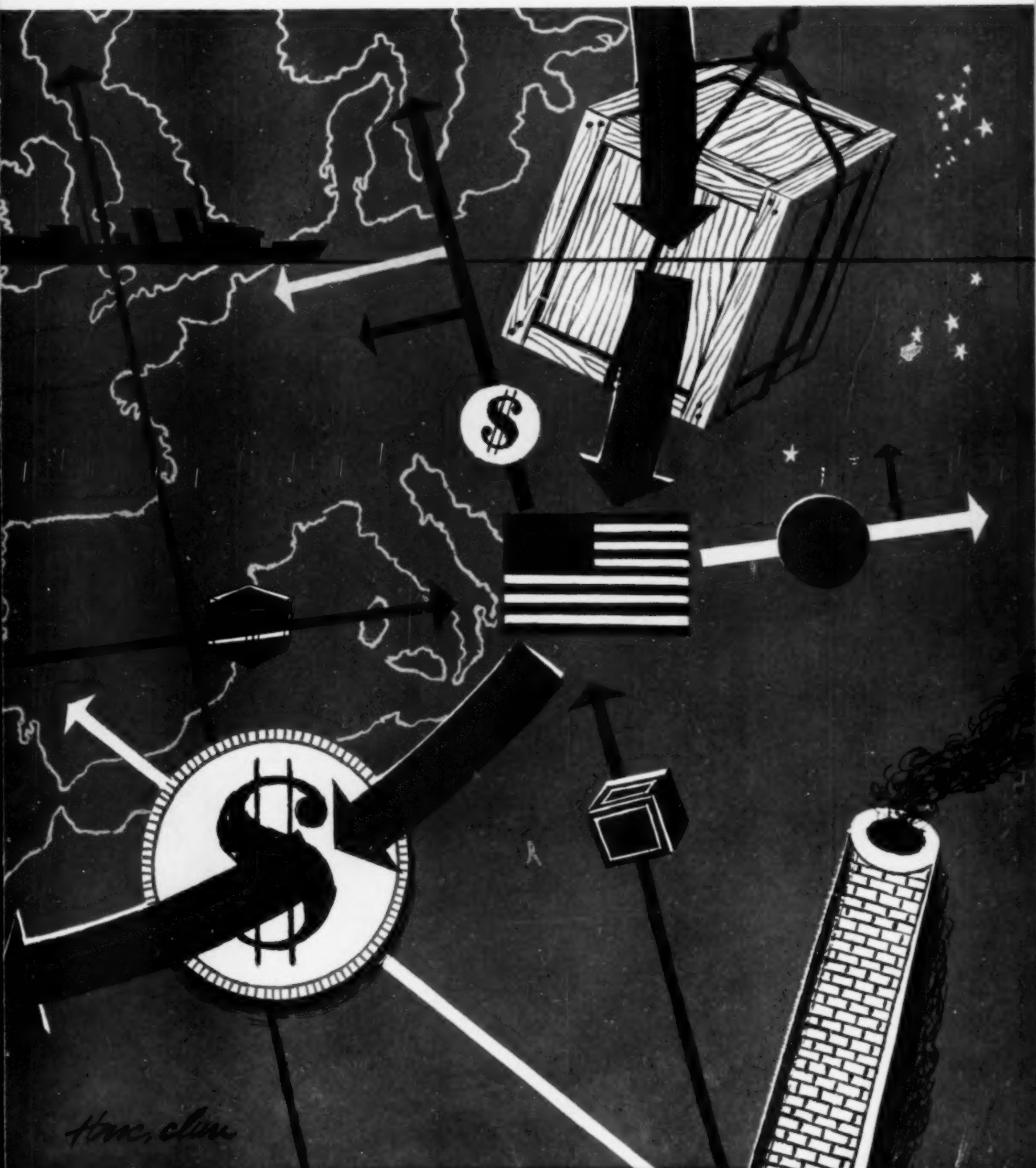
MINING ENGINEERING

OCTOBER 1950



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centrifugal PUMPS

for lead concentrates

For Over 20 Years, this WILFLEY Centrifugal Pump has been operating continuously—handling lead concentrates efficiently with only occasional replacement of wear parts—at a well-known Pacific northwest mill. WILFLEY Pumps are famous for delivering trouble-free performance without attention—stepped-up production and actual dollar savings in power and operation (wherever mill concentrates must be handled economically). There is a highly efficient, cost-reducing WILFLEY Pump size for every pumping requirement. Individual engineering on every application. *Write or wire for complete details.*

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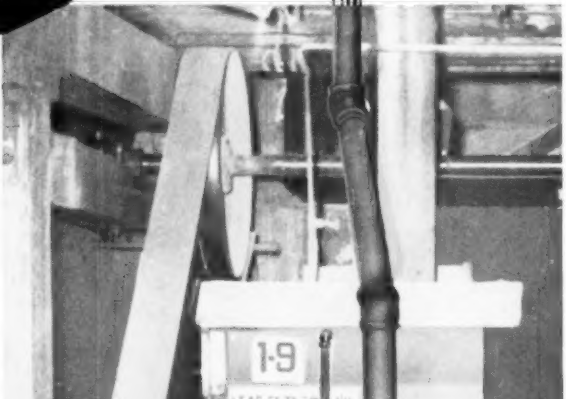

Companion to the famous WILFLEY Acid Pump

MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology
VOL. 187 NO. 10 OCTOBER 1950

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FEATURES

ARTICLES

TRANSACTIONS

Cover: Although our cover this month is a bit abstract, the work of ECA is performing concrete wonders the world over. **See P. 1024 for further enlightenment.**

Engineering Societies Personnel Service

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the four Founder Societies. Local offices of the Personnel Services are at 8 W. 40th St., **New York 18; 100** Farnsworth Ave., **Detroit**; 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York Office and include 6¢ in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter or $12 a year.

POSITIONS OPEN

Assistant Mine Engineer, experienced, to do mine and surface surveying, stope measurement, development work and general mine engineering work. Salary, $3300-$3600 a year. Location, southeastern United States. Y4151.

Mine Superintendent, 30-45, mining graduate, with experience in mining and milling of cassiterite ore, to evaluate primary deposit, plan and supervise shaft sinking and underground operations, prepare production flow sheet, supervise enlargement of pilot mill under direction of general manager. Must have some knowledge of French for reports and correspondence. Two or three year contract as agreed. Salary open. Location, central Africa; 3000ft. elevation. Y4147.

Assistant Manager for a complex metal property, single, 30-40, who has had considerable mining experience, including mine construction work. Must be qualified to assume responsibility and management duties. Salary, $8400 a year. Location, Far East. Y4062.

Draftsman, young, with some mining and metallurgical plant design experience, to be trained for more important work. Permanent position in Southwest with some traveling in Mexico. Y3829.

Division Mine Foreman with underground mining experience, preferably in narrow vein. Salary, $350 a month plus room and board for single man. Furnished apartment at the end of three months if married, plus $65 a month board allowance. Must speak Spanish. Location, Central America, 5000 ft. elevation, moderate climate with dry and rainy seasons. Y3826.

Engineers. (a) Mechanical or Mining Engineers for office engineering work, with some previous experience with belt conveyors, bins, hoppers and bulk material handling. Salary, $3600-$4800 a year. (b) One mechanical and one electrical engineer with some experience in mechanical and electrical layout, machine shop and some design as connected with strip mining. Salary, $5400-$6000 a year. Location, New York, N. Y. Y3783.

Safety Engineer, under 40, with experience in metal mining safety work. Salary open. Location, Michigan. Y4174D.

Chemist-Assayer, graduate, experienced in fire assaying and wet determinations of lead, zinc, copper, gold, silver, tin, tungsten, bismuth, antimony, etc., ores and concentrates, in large laboratory employing 20 workmen and handling about 10,000 assays monthly. Standard three-year contract; working knowledge Spanish essential. Salary, $4200 a year plus one month's bonus. Single status for six months, free transportation to Bolivia by air for employee and wife, free living quarters. Y4186.

MEN AVAILABLE

Placer Engineer, registered, 36, married. Available for exploration work in foreign fields. Capable of following exploration work with plant layout and supervision of mining operations. Experienced in hydraulics, washing plant and dredge operations. Single status during exploration work if necessary. M-581.

Geologist, B. S. One year academic work satisfactorily completed on M. A. Married, two children. Experience as hard rock miner prior to war. Four years in Service. Approximately one year's experience in field on geology of western phosphate. Travel anywhere U. S. M-582.

Mining Engineer and Geologist, 32, graduate French School of Mines equivalent M. S. plus one year post-graduate work mining geology. Fluent French and English. U. S. citizen. Eight years' experience geological mapping and prospection work in France, French Guiana and New Caledonia (gold, bauxite, garnierite, chromite). Seeking position mining geology,

(*Continued on page* 1004)

(Continued from page 1002)

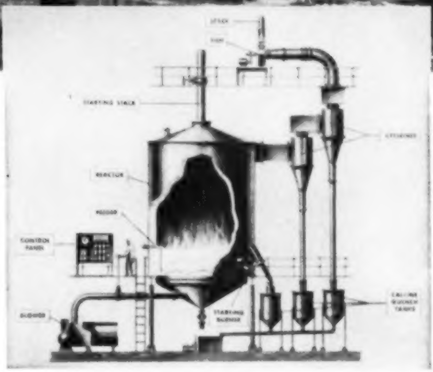
exploration work or laboratory research ore dressing. Available immediately, married, one child. M-584.

Mining Engineer, Dutch (graduate Delft), 34, married, children. Ten years' experience Dutch and American oil and ore companies S. E. Asia and Central Africa. Exploration, evaluation, development (inclusive construction) and management of manganese, iron and phosphate projects. Speaks fluently English, French, German, Malay; reading knowledge Spanish. Interest technical economics, research, sales engineering. Available January 1951. M-587.

Research Engineer, 4 years in charge 70 man laboratory, four years in charge 14 man research department, non-ferrous process metallurgy and mineral dressing; five years mineral dressing research nonmetallics. Age 36, married. Desire position research and development executive, mineral or heavy chemical industry. M-589.

Geologist, 29, single, M. S. Geology, University of New Mexico, 1950. B. S. Agriculture, Ohio State University, 1943. Extensive geologic field work; independent investigations regarding mineralogy and metallography of meteorites, manuscripts covering which are now in press; studies encompassed all phases of geology, including petroleum studies, field mapping and mining methods. Available immediately. Location immaterial. M-590-508-E-8-San Francisco.

Another DORRCO FLUOSOLIDS SYSTEM for Canada's Red Lake district



Circled in the photo above is the FluoSolids building at Campbell Red Lake Mines Ltd. . . . another Dorrco FS System to be installed in Canada for the roasting of refractory arsenopyrite gold ores. With a design capacity of 65 tons per day, Campbell's FS System is roasting flotation concentrates for the elimination of arsenic and sulphur. The calcine is cyanided for gold recovery.

Three features of particular significance have been demonstrated at Campbell:

FIRST . . . the FS System is successfully roasting a concentrate containing only 18% sulphur . . . without auxiliary fuel.

SECOND . . . feed is being pumped to the FS Reactor as a slurry at 70 to 80% solids . . . without prior drying.

THIRD . . . it has been possible to shut down one shift per day with no difficulty experienced in returning to full operation.

Resultant low operating costs and low first cost as compared with conventional roasters make FluoSolids well worth careful investigation. If you have a roasting problem involving arsenopyrite or telluride gold ores we would like to give you further details.

DORRCO FLUOSOLIDS Systems employing single compartment reactors as illustrated are applicable to the roasting of refractory gold ore concentrates, copper and zinc concentrates, and pyrite.

DORRCO

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BARRY PLACE, STAMFORD, CONN.
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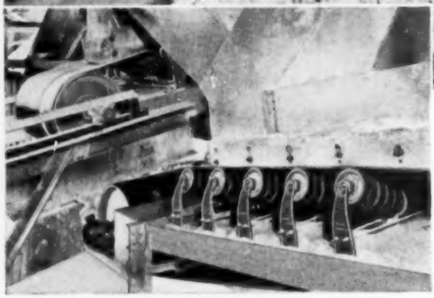
Dorr Technical Services and Equipment Are Also Available Through Associated Companies and Representatives in the Principal Cities of the World. Names and Addresses on Request.

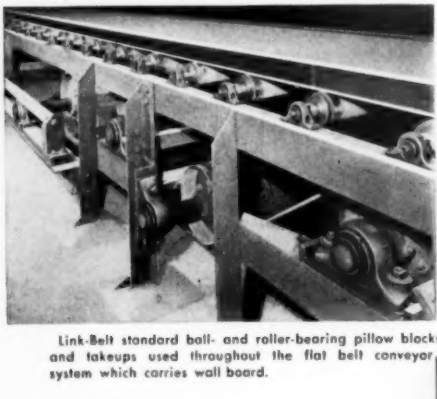
DORR

RESEARCH ENGINEERING EQUIPMENT

A section of a 30" wide Link-Belt roller-bearing belt conveyor handling 500 tons of gypsum rock an hour, showing standard Link-Belt troughed idlers. In the foreground is one of the Link-Belt belt training idlers that prevent damage to the belt from misalignment due to off-center loading, stretch, or improper splicing.



Junction of two Link-Belt roller-bearing belt conveyors showing Link-Belt welded-steel foot pulleys and the rubber-tread impact idlers that absorb the shock of falling material, protecting the idlers and preserving the belt.



Link-Belt standard ball- and roller-bearing pillow blocks and takeups used throughout the flat belt conveyor system which carries wall board.

For materials conveying, whether the problem is tonnage, distance, or a combination of both for any of an almost endless variety of materials, heavy or light, ranging in size from large lumps to fines, for continuous or intermittent operation, it is a Link-Belt job.

Link-Belt designs the belt conveyor system to meet your requirements, builds the various elements, assemblies, supporting structures and enclosures, and installs the job complete. With Link-Belt, experience puts the right equipment in the right place for the specific application and service. Every detail is given the proper attention for trouble-free performance, efficiency, and endurance.

It will pay you to put your materials handling problems up to Link-Belt. Ask our nearest office.

LINK-BELT

BELT CONVEYOR EQUIPMENT

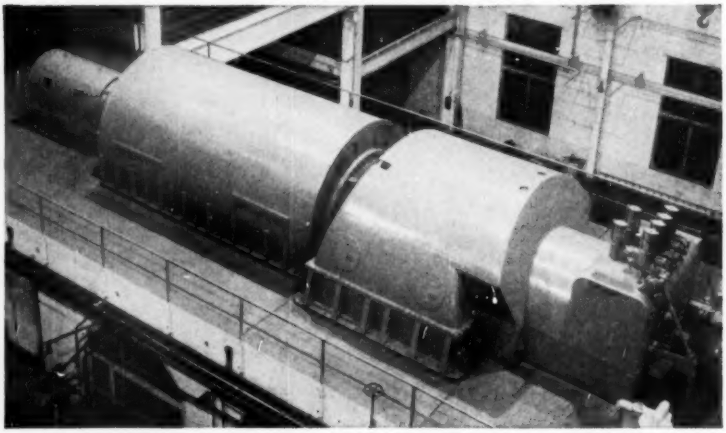
IDLERS • TRIPPERS • BELTS • PULLEYS
BEARINGS • DRIVES

LINK-BELT COMPANY

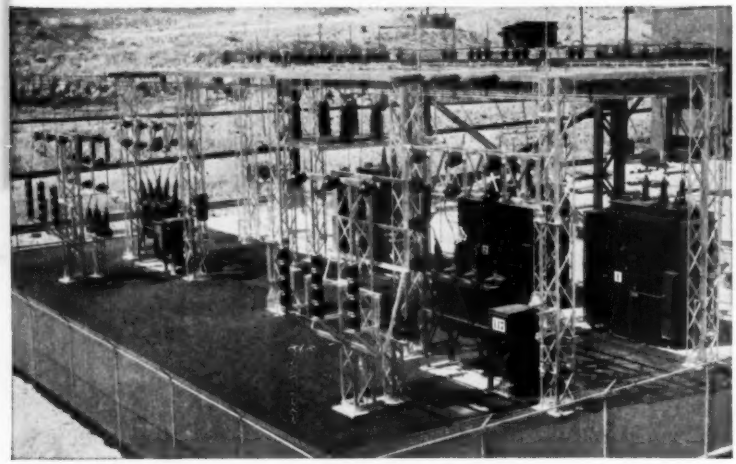
Chicago 9, Indianapolis 6, Philadelphia 40, Atlanta, Houston 1, Minneapolis 5, San Francisco 24, Los Angeles 33, Seattle 4, Toronto 8, Johannesburg. Offices in Principal Cities.

12,051

Powered for low-cost high-tonnage refining!



1 Typical of the mining industry's increasing electrification to handle high tonnages at low cost is this new copper refinery at Garfield, Utah, which has a capacity of 12,000 tons of refined copper per month, with provision for expansion to 16,000 tons. Power for the refinery and other operations comes from Kennecott's power station, where three G-E steam turbine-generators (two of 25,000 kw each, plus the 50,000-kw unit shown here) generate 13.8-kv power which is stepped up to 44 kv for transmission.

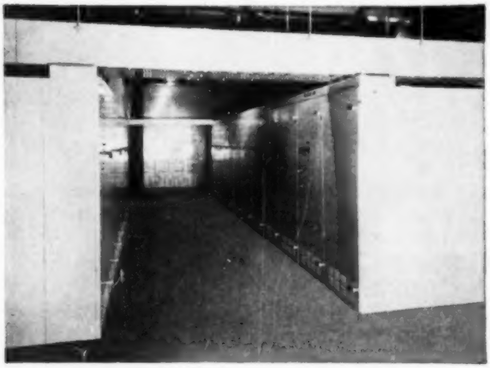


2 At the refinery, three miles away from the power plant, incoming 44-kv power is stepped down to 13.8 kv at this G-E package substation. Completely co-ordinated, it includes an outdoor steel switching structure, three 7500/9375-kva power transformers (with provision for addition of a fourth) and necessary metal-clad switchgear. G-E package substations, made in many standard combinations to fit all mining-industry needs, come in factory-built sections ready to install. They simplify ordering, save engineering time, speed installation, cut costs.

GENERAL ELECTRIC

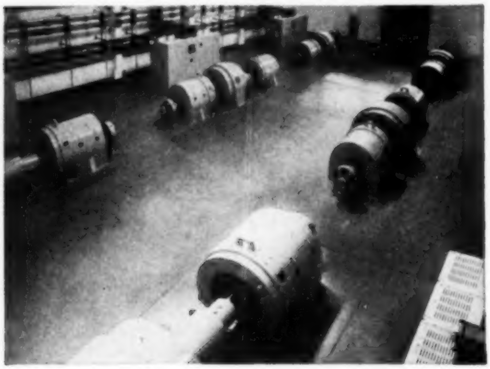
660-20

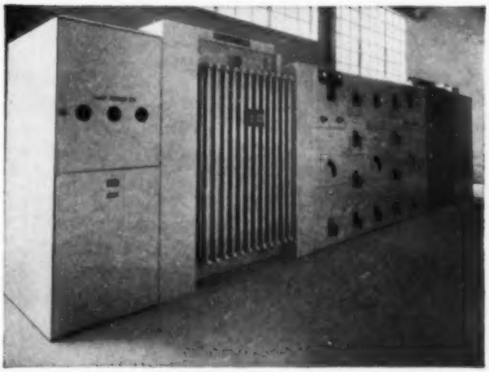
In its new electrolytic refinery, Kennecott Copper Corporation uses this General Electric generation, conversion, and distribution equipment to help maintain production continuity



3 Inside the refinery, protecting all 13.8-kv feeders and personnel as well, is this G-E metal-clad switchgear that provides flexibility for future load changes.

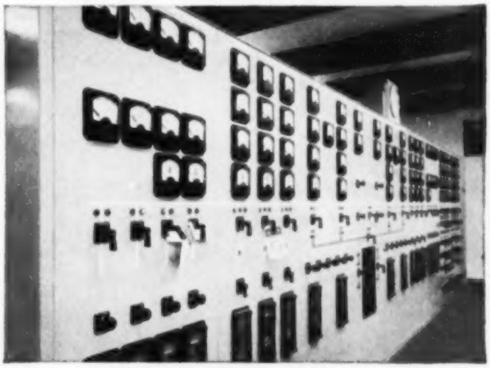
4 A-c is converted to d-c for electrolytic cell lines by six G-E motor-generator sets, each including a 2900-hp synchronous motor and two 1000-kw, 125-volt d-c generators.





5 Four G-E 1500-kva double ended load-center substations (one shown) step down voltage from 13,800 to 480 and provide distribution at centers of load to reduce power losses.

6 This G-E panel — controlling package substation, a-c switchgear, and d-c power for electrolytic cell lines — helps centralize and co-ordinate plant operations.



CO-ORDINATED POWER SYSTEMS

for the Mining Industry

Whatever your mining or processing problem, a good man to know is the mining industry specialist in your nearby G-E office. Ask him about power-system equipment for *your* plant to protect service continuity, help boost production, cut costs. *Apparatus Dept., General Electric Company, Schenectady 5, N. Y.*

Authors in This Issue

A. A. Gustafson (P. 1026) tells us this month how observation of a meat grinder gave birth to a successful "home-made" crusher now used at Freeport Sulphur's operations in Grande Ecaille, La. Now serving as operation's assistant at Grande Ecaille, Mr. Gustafson has been shipping and production superintendent for Freeport in Louisiana, mine superintendent at Nicaro Nickel in Cuba, exploration superintendent in the western states for Freeport, and resident manager of the Freeport Exploration Co. in Canada. He has also been a mining engineer for Pickands Mather in Minnesota. A native of Duluth, Minn., he went to high school there, and to the University of Minnesota for his E.M. degree. He has presented one other paper before the AIME, of which he's a member. Photography, hunting and fishing fill his spare hours.

S. H. Williston

S. R. Zimmerley

S. H. Williston (P. 1037) is vice-president and treasurer of the Cordero Mining Co. in San Francisco. Born in Lawrence, Kansas, he attended Hyde Park High School, the New Mexico School of Mines, and then took his degree in geology and paleontology from the University of Chicago. He served as chief geologist for the Venezuelan-Sun Oil Co., and also did geophysical work for that organization. For the past 15 years he has been engaged in the mining of strategic minerals. His titles include: vice-chairman, National Minerals Advisory Council of the Department of the Interior, and chairman of the alloys committee; member of the executive committee and chairman of the strategic committee of the American Mining Congress; vice-president of the Oregon Mining Assn.; and vice-president and director of the Sperry-Sun Well Surveying Co. Mr. Williston is also an AIME member.

L. E. Schiffman (P. 1047) has spent 23 years with Sloss-Sheffield Steel & Iron Co., Birmingham, Ala., as a mechanical engineer, engaged in raw materials preparation, and now as an electrical engineer. Born in Asheville, N. C., Mr. Schiffman went to high school in Greensboro, and later took his B.S. in electrical engineering from Carnegie Institute of Technology. He spent the first five years of his career with the Springfield Gas & Electric Co. in Springfield, Mo.

J. D. Clark (P. 1068), a native of Columbus, Ohio, took his degree in ceramic engineering from Ohio State in 1939, and for the next four years was associated with Edgar Bros. in Metuchen, N. J. He is now a ceramic engineer with the Foote Mineral Co. in Philadelphia. Mr. Clark now lives in Jenkintown, turns to photography and cabinet making in his spare time.

J. D. Clark

S. R. Zimmerley (P. 1044) holds B.S. and M.S. degrees from Pennsylvania State College and the University of Utah, respectively. Now chief of the metallurgical division of the Bureau of Mines' Region IV in Salt Lake City, he has held various other posts with the Bureau in that city, and has also been an operation and research metallurgist with the Sullivan Mining Co. in Kellogg, Idaho. AIME member Zimmerley has contributed five other papers to Institute literature in the past 22 years.

M. R. Geer (P. 1057), born in Delta, Ohio, attended Ohio State and the University of Washington, receiving B.E.M., M.S., and E.M. degrees. Since his graduation in 1935 he has been a mining engineer with the Bureau of Mines' Northwest Experiment Station in Seattle. He has written seven other papers on coal preparation and utilization for AIME, as well as chapter 2 of the AIME "Coal Preparation" volume. Fishing and woodworking are his hobbies.

Authors

H. F. Yancey (P. 1057) has been with the Pittsburgh Testing Laboratories, and with the Bureau of Mines at Golden, Pittsburgh, Urbana, and Seattle. He headed the solid fuels mission to Germany in 1945, later was with SCAP in Japan, and this year attended the International Coal Conference in Paris. He has presented more than 15 papers before the Institute, on coal preparation and other phases of coal technology. Dr. Yancey attended the University of Missouri and the University of Illinois for his Ph.D. He now lives in Seattle, where he is supervising engineer for the Bureau of Mines.

F. S. Turneaure

F. S. Turneaure (P. 1071) associate professor at the University of Michigan, has been chief geologist for Patino in Bolivia, spent four years as a geologist with the Oliver Iron Mining Co., and five years as a consulting geologist with M. Hochschild, S.A.M.I. in La Paz, Bolivia. He lectured at Harvard in the summer of 1946, and has held his present post since 1947.

O. Hondrum

Olaf Hondrum (P. 1027) was born in Norway, attended high school in Bemidji, Minn., and took his E. M. from the University of Minnesota in 1913. Since then he has worked for Cananea Consolidated as engineer and chief engineer, for United Verde as assistant superintendent and superintendent, and during World War II was assistant general superintendent for the General Chemical Co. He now lives in Bagdad, Ariz., where he's mine superintendent for the Bagdad Copper Corp.

S. F. Ravitz

S. F. Ravitz (P. 1044), professor of metallurgy at the University of California in Berkeley, took his B.A. and M.A. at the University of Utah, and his Ph.D at the California Institute of Technology. He has headed the department of metallurgy and been director at the Utah Engineering Experiment Station, at the University of Utah, headed the metallurgical laboratories section at the Intermountain Experiment Station in Salt Lake City, and, since March of this year, has been at Berkeley.

R. H. Eckhouse (P. 1057) assistant to chief metallurgist for the Southwestern Engineering Corp. in Los Angeles, has been assistant test engineer for Phelps Dodge at Morenci, and a research fellow at the University of Washington. He attended Lawrence Institute of Technology, Western Mich. College of Education, Mich. College of Mining and Technology, and the University of Washington, and holds B. S. and M. S. degrees.

* As part of $850,000 granted by the House Appropriations Committee for development of manganese in the U.S., the Bureau of Mines will build a pilot plant at Artillery Peak, Ariz., for development of low grade manganese ores.

* The historic Argo tunnel and mill in Idaho Springs, Colo., will reopen shortly for the mining of uranium. The Uranium Co. of America has acquired control of the tunnel and mill and 12 nearby mining properties. J. H. Rodgers and G. C. Ridland will manage the operation.

* The opening of a new 4000-ton per day coal mine in Washington Co., Pa., and the reopening of another mine of the same capacity in Fayette Co. has been announced by the H. C. Frick Coke Co. They'll help meet added war demands.

* A 7000 bbl per day commercial synthetic liquid fuel plant, making gasoline from natural gas at Brownsville, Texas is being operated by the Carthage Hydrocol Corp., a Texaco subsidiary. The gasoline is produced by the Hydrocol process, chemically similar to the Fischer Tropsch process, but differing mechanically in several ways. To operate the process it has been necessary to build an oxygen plant producing 95 pct pure oxygen in quantities equal to U. S. production of commercial oxygen in 1946.

* Over 96 pct of the class of 1950 at Montana School of Mines are now employed, in industry, and they're scattered from MIT to the island of Cuba.

* Cobalt, heretofore almost entirely imported, will soon be produced in Idaho. The Calera Mining Co. is building a 600-ton mill in which a cobalt and copper concentrate will be obtained from Idaho ores.

* Plans for mining iron ore on the Conarky Peninsula in French Guinea, (an ECA project) call for equipment purchases as follows: 8 wagon drills, electric shovel for mining, conveyor for the port, 8 trucks, wrecker, 2 tractors, as well as miscellaneous accessories and spare parts.

* A low-grade lead-zinc mineralization, located east of any known ore bodies in Utah's East Tintic district, has been located by a 2000-ft wildcat USGS drill hole.

* Higher-than-going rates have been authorized by the Munitions Board for its stockpile program, where domestic production of scarce materials is low or uneconomical. Payment in excess of 25 pct over the foreign price is now permitted.

* A new self-centering conveyor roll, developed by Carnegie-Illinois Steel, is expected to revolutionize materials handling. It completely eliminates the necessity for sideguides to keep materials on a conveyor. See P. 1028.

* Great Britain's socialized coal industry employs twice as many miners, but produces less than half as much coal as America's coal industry. Last year, British mines produced 28 million fewer tons of coal than the output of the same mines in 1939. Absenteeism amounted to a big 12 pct of working time.

* Labor market conditions have tightened on almost a country-wide scale. Only 4 out of 143 major labor market areas have a substantial surplus -- less than half the total for May. Growing scarcities of skilled labor are in evidence.

Taking the Long View

THE present international situation has serious implications for this nation. The possibility of total war is no more appalling than the threat of making the United States permanently into an armed camp. A period of ten years under such conditions could mean a Communist victory without firing a shot—if the present trend toward statism is allowed to continue.

As individual citizens and as members of the mining profession we have the power to oppose the march toward socialism. We must not adopt short-sighted policies dictated by self-interest, but must endeavor to size up the situation as a whole and act according to what will be best in the national interest.

The economic picture for the metal mining industry under present emergency conditions is apparently bright. Demand for metals exceeds supply. But there is danger that fluctuating prices will bring Government regulation. Care must be exercised in our pricing policies, or the calamity of Government intervention will be inevitable.

President Truman's "loophole" reference to percentage depletion early in the year caused a brief flurry of excitement, but now that the immediate threat is over the industry has lapsed into sommolence on this subject. Special tax privileges accorded the mining industry because of inherent risks should be guarded by a continuing propaganda campaign to educate Congress and the public. If this is not done we may find ourselves losing the necessary tax protection which we now possess by a capricious act of legislation.

Stockpiling minerals from abroad certainly lengthens the life of our own mineral resources. The present consumer demand for the base metals indicates foreign sources of supply for stockpiling. However, this is only one facet of the problem. For strategic military reasons it is necessary that the United States have quick sources of all minerals within our national boundaries. Undeveloped mineral deposits do not meet this need so that where it is possible to encourage our own mineral development by stockpile purchases, it should be done. A need exists for an overall controlling agency that would make purchases for the stockpile according to military necessity, current metal market conditions, and so as to strengthen the mining industry.

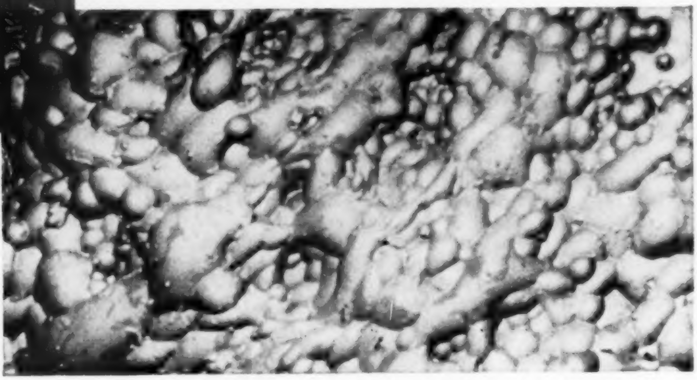
In the course of the history of the United States we have been engulfed in several wars by the necessity of protecting our economic rights abroad—trading privileges—and also because of moral obligations. Disregarding the moral side, it is difficult to reconcile fighting wars to protect world freedom of trade when in peacetime we attempt to restrain such trade by tariff barriers.

When it comes to accepting subsidies from the Government in one form or another it is difficult to overlook the immediate personal gains and see in the future that where the Government lays out capital it becomes a partner in the business and not a silent one.

It is apparent that hope for the traditional American way of life cannot be bought without some self-sacrifice; and we cannot wait for the other fellow to start it because it is the responsibility of each individual citizen. By taking the long view our course should be apparent and not diverge sharply from our own personal benefit.

Dependable Bear Brand Xanthates

"Over Twenty-Five Years' Experience in Producing Xanthates for Metallurgical Use."



BEAR BRAND XANTHATES AVAILABLE

- **Z-3—Potassium Ethyl Xanthate**
- **Z-4—Sodium Ethyl Xanthate**
- **Z-5—Potassium Amyl Xanthate***
- **Z-6—Potassium Pentasol Amyl Xanthate***
- **Z-8—Potassium Sec.-Butyl Xanthate**
- **Z-9—Potassium Isopropyl Xanthate**

*From Sharples Amyl Alcohols

Time and experience have demonstrated that for optimum results in the flotation treatment of substantially all sulphide ores, as well as some oxidized ores and ores containing native metals, Bear Brand Xanthates are the cheapest and most efficient collectors now available.

GREAT WESTERN DIVISION
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THE DOW CHEMICAL COMPANY

DOW

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San Francisco 4, Calif., U. S. A.

It's Everyone's Business

Republic, Armco, Buy Reserve Mining Co., Plan $60 million Taconite Plant

REPUBLIC STEEL CORP. and Armco Steel Corp. have joined in a $160,000,000 project for the production of iron ore from Taconite in the Lake Superior mining region. The two companies announced acquisition in equal shares of 100 pct ownership of the stock of Reserve Mining Co., which controls a vast deposit of magnetic Taconite iron ore on the eastern end of the Mesabi range in Minnesota.

Through Reserve Mining the two companies will build a plant for manufacturing high grade ore from Taconite near Beaver Bay on the north shore of Lake Superior. The first announcement calls for about a $60,000,000 plant which will be built as soon as plans are completed and will have an annual capacity of about 2,500,000 tons of iron ore pellets. Longer range development plans call for future expansion of the plant to provide an annual capacity of 10,000,000 tons. This will amount to an additional investment of $100,000,000, the company said. Financing and actual production plans were not disclosed. An organization meeting of Reserve Mining under the new ownership is planned for next month, when financing and other operating details will be worked out.

Armco had held one third interest in Reserve Mining with Wheeling Steel Corp. holding one third and the remainder being held by Cleveland-Cliffs Iron Co. and Montreal Mining Co. Republic Steel has now acquired the former holdings of Cleveland-Cliffs and Montreal Mining and half the interest of Wheeling. Armco has acquired the other one half of Wheeling Steel holdings—thus, each of the companies holds a half interest in the new Reserve Mining Co. structure. Oglebay Norton & Co. of Cleveland, who has managed Reserve Mining properties, will continue in that capacity.

American Mining Congress Convention

A GENERAL attitude apparent at the recent American Mining Congress Convention in Salt Lake City was a revitalized spirit among representatives of the mining industry. Undoubtedly this healthy feeling is due to the anticipated strategic role the domestic mining industry will be asked to play in plans for mobilization. Certainly a new spirit is highly desirable if the industry is to shake off the restricting bonds of pessimism which were fashioned by developments in the recent postwar years.

It is to be hoped that the rising tide of optimism will aid in mitigating disunity among the various segments of the industry. Lack of unity is a primary reason for the failure of the mining industry to present adequately its case before the nation to the end that the industry may command more attention in the national economic picture. In order to become a potent economic and political force each group representing certain related metals or minerals should aid other groups.

A spirit of cooperation was no better exemplified than at an informal meeting of the secretaries of various mining associations on August 27 just prior to the convention. It was the consensus of that group that each mining association needs the aid of others because in most if not all states the mining industry is in the minority. At this meeting most were agreed that the mining industry is in need of a large public relations organization to reach the general public.

Congressman Carl T. Durham (N.C.), Chairman of the House Armed Services Subcommittee on Stockpiling, told attendants at the convention in a speech on August 29, that in the long run it is probably sounder policy to acquire cheaper metals and minerals for the stockpile from abroad, notwithstanding the fact that development of domestic deposits suffers. He stated that the "more we get from abroad now, the longer we defer exhaustion of our own supplies in the ground." On the other hand, Congressman Durham emphasized that the short run point of view dictates that "it is essential that we develop and preserve enough minerals producing capacity to provide us with the materials we need for national defense and to discharge our responsibilities as a leading member of the United Nations." He said that "Our policy of buying cheaper from outside the United States may turn out to be the most expensive mistake in our history."

Discussions on strategic metals and minerals led to the conclusion that there are large low-grade deposits of manganese, chrome, tungsten, molybdenum and others in this country which could be utilized in an emergency with some form of price support, liberalized specifications in some cases and financial help where most needed. Most strategic metals produced domestically cannot normally compete with foreign supplies.

National Minerals Advisory Council

At a meeting of the National Minerals Advisory Council on September 1 at Salt Lake City the seven Commodity Committees submitted reports on the various metals and minerals. Contents of the reports covered production estimates in the years immediately ahead, steel requirements, estimates of military, civilian and stockpile requirements, transportation problems, etc., and recommendations of methods and procedures to be followed which, in the opinion of industry experts, will achieve the desired results in preparing for adequate defense.

The Council adopted a resolution on the manpower problem and made recommendations as follows: "BE IT RESOLVED that (1) the Council recommends that the manpower policies of the Government, with respect to the draft, the activating of reserves and National Guards, and in other respects, should be framed in the light of the need for minerals * * * and (2) specifically, the Council recommends that the list of critical occupations, issued by the Secretary of Labor on August 3, 1950, for the guidance of the Department of Defense in expanding the Armed Forces, should be amended to include miners and other skilled operating, supervising and management personnel in the mining and metallurgical industries."

Council recommendations at the August 3 meeting have had some effect. At that meeting the Council adopted a resolution on the manpower problem somewhat more general than the one above (see September issue) and the Dept. of the Interior immediately assigned a staff to work on the problem. At the same meeting the Council urged that Section 209 of the tax bill (H.R. 8920) relating to abandonment of capital assets and certain property used in the trade or business be stricken from the bill. The Dept. of the Interior subsequently sent a letter to the Secretary of the Treasury favoring the deletion of this provision. Section 209 has been stricken from the tax bill.

← French coal miners studying a new shaker conveyor purchased with ECA funds.

The Pattern of ECA in Mineral Affairs

By C. H. Burgess

ON June 5, 1947, Secretary of State George C. Marshall in a speech at Harvard University outlined a plan for the economic recovery of Europe. The plan contemplated that the United States should provide assistance which would enable the European nations better to help themselves and to help each other to reconstitute from their war-torn lands conditions which would make possible decent standards of living.

The Foreign Assistance Act of 1948 was passed in April of that year, and established the Economic Cooperation Administration and set United States policy on the aid program. Since the passage of the Act, economic recovery in Europe has made remarkable progress. The total output of goods and services in 1949 was 25 pct above that of 1947, and exceeded even the pre-war level; the expansion is continuing and a further increase of about 10 pct should be possible by June 30, 1952, when ECA comes to an end. Both in industry and agriculture recovery has gone much faster and farther than in the four years after the first World War. In anticipation of the time when the ECA assistance will no longer be forthcoming the Marshall Plan countries are striving to improve their dollar position by increasing their exports to the United States and by reducing their imports from the United States.

Mr. Burgess is Director, Strategic Materials Division, Economic Cooperation Administration, Washington, D. C.

The Economic Cooperation Administration is a financing agency. Its funds are not given away to business men or farmers or other individuals who need dollars for purchases, but are made available only to the governments of the receiving nations. I would like to give a hypothetical case which demonstrates how the scheme works. Let us assume that a French farmer wants to buy a tractor of American manufacture which costs $2,000, and let us assume further that the tractor is one of the items which both the French Government and ECA have determined to be suitable objects of Marshall Plan financing. The farmer then deposits with the French Government a sum of francs equivalent in value to the $2,000 price of the tractor, and ECA provides the dollars which the United States manufacturer of the tractor receives. The francs, which are the counterpart in value of the dollars granted France by ECA, are held in two special accounts in France. One of these accounts contains 95 pct of the franc equivalent of the $2000. This 95 pct counterpart fund belongs to the French Government, and may be used for debt retirement, public works, or loans to private firms, but these funds can be spent by the French Government only with specific approval of the Economic Cooperation Administration. The remaining 5 pct of the franc counterpart of the $2000 is deposited in a second special account, and belongs to the United States Government. It can be used only for the administrative expenditures of the United States Government, such as rentals of Embassy buildings and various services in France, and for the development and acquisition of strategic materials originating in France or marketed there.

Several aspects of the Economic Cooperation Administration hold special interest for the mining industry. You may not be fully aware that from April, 1948 to July, 1950 purchases with Marshall Plan funds were authorized for a monthly average quantity of 24,000 long tons of *copper* and *copper products;* for 5,100 tons of *lead* and *lead base alloys;* for 9,350 tons of *zinc* and its *alloys;* and 13,500 tons of *aluminum.* The total of non-ferrous metals and products whose purchase has been authorized in this period comes to 1,425,000 tons having a value of $576,215,000. In the same period the purchase of non-ferrous ores and concentrates worth $107,134,000 has been authorized, and *machinery* and *vehicles* valued at $1,552,900,000.

A desire to get something back in return for Marshall Plan aid, plus the awareness of the accelerated wartime depletion of United States mineral resources, a subject which had received widespread public notice, caused the Congress

to include in the original legislation specific provisions concerning acquisition of strategic materials for stockpiling in the United States.

The scope of the Act is broader than the stockpiling concept, referring to "materials which are required by the United States as a result of deficiencies or potential deficiencies in its own resources," and it expresses the intention to assist American industry in obtaining foreign supplies of these materials. ECA is charged with responsibility to promote their production.

The activity of the Economic Cooperation Administration with respect to strategic materials has proceeded along two courses. One is the acquisition by purchase with the 5 pct counterpart funds whose derivation has been explained earlier. In most cases these acquisitions have been made as spot purchases at current market prices. The commodities which have been obtained include rubber, sisal, diamonds, bauxite, palm oil, and other commodities, which, as they are destined for the stockpile, must meet the Munitions Board specifications.

Although the 5 pct counterpart funds which have accrued from the inception of ECA to June 30, 1950 total the equivalent of about $272,600,000 the equivalent of only $61,370,000 have been employed in purchases of strategic materials to September, 1950. Inability to utilize these funds more fully is accounted for by several factors. One is that in the case of some countries such as Greece, Italy and Austria very large sums of 5 pct counterpart funds have accumulated but the countries are so poor in natural resources that they have no strategic materials surplus to their own needs. The opposite extreme is best exemplified in the case of Belgium where there have been virtually no 5 pct counterpart funds with which to purchase strategic materials although the Belgian Congo is a producer of large surpluses of these materials. (The reason why these funds have not been larger in Belgium is that Belgium has for the most part received loans rather than grants, and the counterpart funds accrue only in the case of the grants.) An additional factor has been that in the occupied countries on the continent industrial stocks were almost completely exhausted during the German occupation.

The second major type of ECA activity in the field of strategic materials consists of financing projects designed to explore for, develop, or produce strategic materials. The sums obligated for these projects to September, 1950, total $17,903,000 plus the equivalent in the 5 pct counterpart funds of $18,049,000, or a total of $35,952,000. Although one of the survey projects is being carried out by the foreign government, all the others are carried out by private operators. You may have seen notices of several undertakings on which contracts have been executed within recent months. One consists of an advance of $3,600,000 plus francs from the 5 pct counterpart funds equivalent to $4,000,000 for the purchase of mining and milling equipment to be installed at a lead-zinc mine in French Morocco. This money will be repaid with simple interest at 4 pct in lead and zinc which will be delivered to the United States stockpile. The recipient of the funds will be credited for the metal delivered at the New York market price current at the time of delivery after deducting United States import duty, ocean freight and insurance. Thus the United States will receive full return for the dollars and the francs it advances. The contract also provides that the Federal Supply Service, which is the chief procurement agency for the United States stockpile, will have an option to purchase a fixed percentage of the output of this property over a term of years following the re-payment of the ECA advance. The increased output which will result from the installations which are being financed in part by ECA and in part by the private owners of the property will enable France to become nearly self-sufficient with respect to lead and zinc.

A second contract concerns the supplying by ECA of both dollars and pounds sterling to purchase bauxite mining, drying and loading equipment to be installed by a United States mining company on its property in Jamaica. In this instance repayment to the United States of principal and interest will take the form of aluminum metal of stockpiling grade. For the purpose of crediting the company the metal will be accepted at the New York price prevailing at the time of delivery. This installation will have great strategic significance in utilizing a source of bauxite 1500 miles closer to the United States mainland than the sources in the Guianas.

A third contract which has been executed recently departs from the two described above in being an advance to a government instead of to a private party. In this case the Government of Sweden may draw up to $350,000 for the acquisition of equipment of United States manufacture for the expansion of output of lead-zinc mines more than 1000 years old. In this case the funds will bear interest at 2½ pct instead of 4 pct, the lower rate being granted because the repayment is guaranteed by a government instead of by a private firm. In this case repayment may be made in dollars rather than metal if Sweden requires the metal for its own use at the time the installments of repayment fall due.

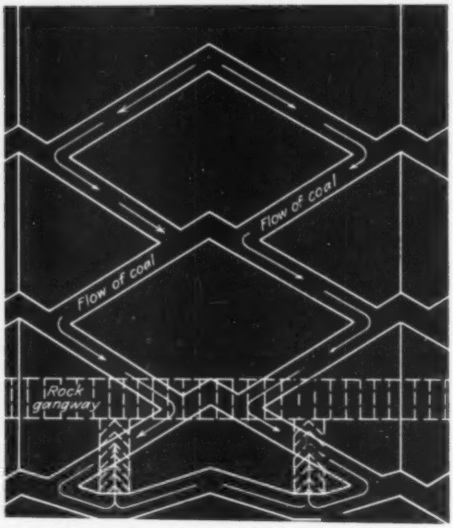
In the cases above cited which are described as advances against production the parties operating the mines have already made substantial capital investments and will continue to do so in these programs. The repayment of the ECA funds will proceed according to a negotiated schedule, with the recipient having the right to pay back at an accelerated rate.

Perhaps the most striking feature of our experience with respect to the development projects has been the apparent lack of interest on the part of American mining interests to seek ECA assistance. Some of the reasons appear to be fear of political instability, nationalization of industry, and excessive governmental interference in the country or territory where the venture would be conducted; paucity of good prospects and mines and poor transportation; adverse provisions of the United States income tax rules; the belief that coping with ECA red tape is not worth the candle; and lack of assurance of a market for a large portion of the prospective new production over the period of amortization. I should like to take this occasion to inform you of our willingness to consider applications for ECA participation in the exploration or development of new properties.

Techniques for

Pitch Mining *in* Anthracite

by Garfield A. Schnee



The diamond shape method, used for 5 years at Philadelphia & Reading's Knickerbocker colliery, has 14 pct greater recovery than conventional methods.

A Pennsylvania Anthracite Section Contribution

MACHINERY has not taken the place of manual labor in steep pitch coal in the anthracite field and there is a shortage of miners experienced in this type of work. To overcome these difficulties several coal companies have resorted to different types of mining such as the diamond or zig-zag method, the slant method, long hole drilling, and several other variations.

For the past 5 years the diamond shape method has been used extensively in the heavy pitching beds of both the Knickerbocker and Yatesville basins at Knickerbocker colliery of the Philadelphia & Reading Coal & Iron Co.

In this method of mining, development is made from a gangway in or underneath the vein. From the gangway, chutes or rock holes are driven on 60-ft centers and are connected by a slant monkey airway. Slant chutes are then driven from the monkey airway along the bottom slate on 30° across the pitch for a distance of 60 ft. After dropping back 2 sets from the face, another slant is started above the top of the timber and driven in the opposite direction along the bottom slate on 30° for a distance of 60 ft. This method of development is continued until the zig-zag slant chutes are driven to their upper limits.

Mr. Schnee, an AIME member, is division superintendent, Philadelphia & Reading Coal & Iron Co., Ashland, Pa. This paper was presented before the Anthracite Chapter meeting on April 28.

When mining in virgin coal, the slant chutes driven from the adjacent chutes or rock holes intercept at the apex of the diamond shaped pillar. This method is also applied in recovering pillars in areas which have been first mined. The length of each slant has to be modified and the reverse in direction made when the slant holes into the rib of the breast.

Timbering in the slants is done with two piece post and bar sets of 7 or 8-in. timber and a center prop is used to split the slant into a chute and manway. Sheet iron is placed on the bottom of the chute section which is along the bottom slate and ventilation is carried in the manway section. Air batteries, with regulators, are installed at the apex of the slants.

In the second mining, one half of the diamond shaped pillar becomes tributary to each slant off each working place. The slant is robbed in two or three sections. Holes of various lengths up to a maximum of 45 ft are drilled in the pillar parallel to the bottom slate of the vein and each section is brought back 20 to 25 ft. If the vein is thick, a back hole is driven to the top slate and additional long holes are drilled in the top coal. Sometimes, in order to keep the drill holes to a maximum of 45 ft, it is necessary to drive narrow width holes up the pitch a short distance off the slant from where holes are then drilled to blow the top coal.

All holes are drilled with an air driven twist steel drill equipped with 2¼-in. fish tail bits. The holes are charged with 1½-in. coal powder and a delay detonator placed in the center of the charge. In each hole Primacord is also placed along the entire length. The holes are fired by electricity and the charge detonates the Primacord which in turn detonates any powder which may not have gone off in the initial blast.

From accurate records, this system of mining has a recovery of 14 pct greater than from an area in the same vein where the conventional breast and pillar method of mining was done.

Slant Chute Method—Germantown Mine

The slant chute method of mining has proved to be the most satisfactory of the various methods which have been used in mining the Mammoth vein at Germantown mine of the Raven Run Coal Co. The method used is similar in

many respects to the one used at the Newkirk colliery of Philadelphia & Reading.

The vein is approximately 25 ft thick with a slate top and bottom. It is gaseous and has a pitch which varies from 45 to 80°.

The vein is developed by means of an 8x11 ft rock gangway or Skidmore vein gangway with 5x9 ft rock holes approximately 25 ft long on 45° pitch. Rock holes are on 60-ft centers where the coal is very hard and on 70-ft centers where the coal is slippy.

A manway, chute and battery is constructed in the rock hole and a 5x9 ft back chute on 30° is driven through the vein to the top rock. Main air headings 5x6 ft are then driven east and west along the top rock to connect with the back chutes from adjoining rock holes. Booster fans and ventube are used to provide air in the main headings until they are connected and positive ventilation established.

A 6x10 ft slant chute on 35° pitch is then started up along the bottom slate and advanced 40 ft in the direction in which the gangway is being driven where it is met by a cross-heading which has been driven from the main airway. Double 6-in timber on 5-ft centers is used in the main air headings and the slant chutes with a center prop in the slant to divide them into a chute and manway.

The air current is directed through the cross-heading and up the manway of the slant to a point where it is turned into the coal chute by means of a stopping. From this point it travels up the coal chute to the face, then down the manway to the cross-heading and up through the cross-heading to the next slant. Cross-headings are driven on 50-ft centers.

After the slant chute has reached its limit, a back hole is driven to the top rock and robbing is started. The pillar between the slants which is 36 ft, is removed by retreating 25 ft at a time down the slant chute. Progress of the robbing is controlled so that the miners in the overlying chute are not endangered by those working underneath which also insures a uniform retreat of the robbing face. Engineers check the pitch of the slant chutes, keep the surveys close to the face and construct cross-sections through the pillar at each cross-heading to guard against one chute being driven into another.

Germantown mine officials favor this method of mining because the light pitch of the chutes makes the mining less hazardous and the work much easier. Also, less experienced men are required and older miners who can no longer work on heavy pitch may be used. Further, there is less breakage in the coal running down the chutes; lower costs in chute repairs; bad bottom or rushes of rock in the open cut can be controlled and from actual records, the cars per man-day have increased over the pitch method.

Slant Chute Method—Newkirk Colliery

The slant chute method of mining heavy pitching veins at Newkirk colliery of Philadelphia & Reading is similar in many respects to that used at the Germantown mine. The veins are on 75 to 85° pitch and are 8 to 10 ft thick.

Development is made by either a gangway in the vein with chutes on 60-ft centers or by rock gangway with rock holes on 60-ft centers. The airway is driven on the strike of the seam at the top of the rock holes or coal chutes. Slant chutes are then driven on 45° across the pitch with headings at right angles on 60-ft centers. Timbering is done with post and bar sets with a center prop to split the slant into a chute and manway. Sets consist of a 7-ft bar and a 7-ft post with an 8-ft spread at the bottom.

After the slants have been driven to their limit or holed into the level above, the mining of the pillars is started. A chute battery is erected across the slant at approximately 25 to 30 ft below the face depending upon the quality of the coal, the character of the top and bottom slate, and the thickness of the seam. Narrow panels and short lifts or skips off the slant chutes favor the coal recovery and reduce the dilution.

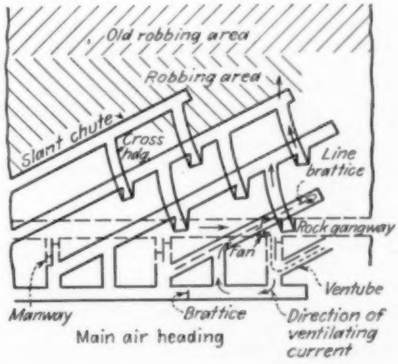
After the first 30 ft have been mined, another battery is erected 25 to 30 ft below the first battery and the same procedure in mining is repeated until all of the coal has been mined to the air heading.

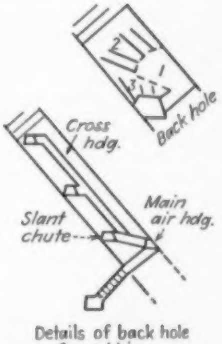
Two miners have produced 54 150-cu ft cars by this method in one shift of 7 hr and approximately 80 pct of the vein has been recovered.

Long Drill Holes

The use of long drill holes is becoming increasingly popular in studies for different types of pitch mining. One of the major objections to the long holes was the inability to control the drift of the holes. This presented the danger of a hole

The slant chute method as used at the Germantown mine of the Raven Run Coal Co. The 35 pitch of the chute has made mining less hazardous.





Details of back hole for robbing

blowing into an opening where there may have been an accumulation of gas. Another objection was the possibility of some of the powder burning in a hole and causing fire. While these objections still remain, present day improvements in drill equipment with more rigid rod connections have somewhat reduced the amount of drift in a hole, also, the use of Primacord for the full length of the hole as explained previously under the diamond shape method has lessened the possibility of burning powder.

Experimental Blasthole Mining

Interest in long hole methods has been revived at the Lehigh Navigation Coal Co. and as a result, an experimental section has been set aside and drilling equipment procured to conduct further development work on the use of long holes for mining anthracite. However, no definite conclusions can be herein presented.

Many plans for utilizing long holes to mine the vein in the experimental section were prepared. In general, the plan provides for the driving of an initial opening in the vein to cut the coal from the bottom slate to the top slate and to extend up the pitch to the gob above. Only sufficient coal is to be drawn from the initial opening to provide for the expansion of the material when it is blasted. Drilling will then be done in a vertical plane along the rib of the initial opening so that the holes can be charged and blasted to slab off or break the rib. Drawing of coal from the breast and from the necessary draw holes along the bottom of the vein will then be done to compensate for expansion of the coal and to up-set the particles of coal just blasted in preparation for succeeding blasts. Successive drilling, blasting and drawing steps can then be followed to mine the vein.

The drill hole pattern for this mining can be of the fan or parallel hole type. Fan hole drilling involves holes across various layers of coal some of which may be soft or free and cave, thus blocking the holes. Fan hole drilling, however, requires the minimum number of drill setups and the least amount of development work for placing all of the drill holes. One unique plan being investigated is to drill fan holes from a drill heading under the vein. This method has the advantage of keeping the miners out of the coal entirely and minimizes development work.

Parallel holes can be drilled from cross holes driven from the bottom rock to the top rock at the desired interval along the vein. In most of the plans prepared, these openings are to be driven directly off the rock chute taps to the vein which after blasting will serve as draw points for loading the broken material.

Drilling is to be done with a post mounted blasthole diamond drill powered by an air motor. Noncoring bits of either the pilot or plug type will be used. This type drill and bit was selected for the initial test work because it is capable of cutting any type rock or coal with the least amount of deflection. Should the water used to remove the cuttings from the hole cause trouble when drilling the free vein material, an auger drill rod with a tungsten carbide bit will be used.

The present plan for blasting provides for the use of a semigelatinous permissible powder. Detonation of the charges in the various holes is to be done with millisecond delays to reduce the concussion from the blast. At least two detonators per hole will be used in an attempt to avoid cutoff holes. The use of Primacord in the holes and millisecond delays to detonate the Primacord is also being considered.

Experimental work is now at the point where miners are being trained in the use of the blasthole drill. Holes are being drilled to learn the drilling characteristics of the vein to be mined and the drill pattern to be used will depend on the results obtained from the test drilling.

Breast and Pillar Mining Using Long Drill Holes

One modified method of breast and pillar work utilizing long drill holes is under consideration at the present time at Locust Gap colliery of Philadelphia & Reading. (See cut, p. 1025 D)

The plan provides for the development of the three splits of the Mammoth vein, which range in thickness from 6 to 17 ft and are on a pitch of 70 to 85°, by driving a haulage gangway and return airway in the Buck Mountain vein. This haulage gangway and airway will eventually be 5000 ft long. Sectional tunnels will be driven from the haulage gangway to the top member of the Mammoth vein on 360-ft intervals. Midway between the sectional tunnels, an air tunnel will be driven at a point approximately 100 ft higher than the gangway.

Instead of driving gangways in the vein from the sectional tunnels, slants will be driven along the bottom slate of the vein on 22° pitch for a distance of 192 ft where they will meet the slants driven from the adjoining sectional tunnels. At the junction point of the slants, a chute will be driven up the pitch to intercept the air tunnel.

After the air connections are made, twin breasts, 20 ft wide on 40-ft centers are to be driven at the sectional tunnel and at the end of the slants, one on each side of the tunnel and one on each side of the apex of the slants.

The breasts at the tunnel will be driven up the pitch for a distance of 190 ft and at the apex of the slant, a distance of 130 ft, and will carry a full gob with a 3-ft manway on each side. From the face of the breasts, approximately 700 ft of $2\frac{1}{2}$-in. holes will be drilled ahead and on the flank with an air driven rotary drill, to drain off any water which may be impounded in the completed workings directly above. After the drilling has been completed, narrow holes 10 to 12 ft wide will be driven the remaining 50 ft to the gangway above. Headings between the breasts are to be on 60-ft centers but those on the solid side are to be on 30-ft centers and driven 26 ft into the 120-ft block of coal.

When the driving of the breasts has been completed, 3 spouts on 30-ft centers are to be driven off the slant for a distance of 10 ft to permit the construction of breast batteries with short 3-ft manways on each side.

Holes $1\frac{3}{4}$ in. in diam will then be drilled from each spout with an air driven rotary drill on a planned depth of from 6 to 30 ft. The total footage for the first step will be approximately 235 ft. The holes will then be loaded with permissible explosives and fired with instantaneous, first delay and second delay exploders. This plan provides for an 80-ft face from the breast headings

in no. 1 breast to the breast headings in no. 2 breast. Sufficient coal will have to be drawn from the three spouts to lower the loose coal from the face to permit circulation of air and to permit the expansion of the coal which will be blasted in the succeeding step.

The second step provides for drilling the 80-ft face from the first heading off no. 1 and no. 2 breasts. Planned depth of the holes is from 20 to 60 ft with a total requirement of approximately 290 ft. The holes will then be loaded and fired as previously described. The same procedure will be followed in the succeeding steps until the block of coal has been mined which will require approximately 1975 ft of drill hole.

Mining of the 20 ft pillars will be done in proper sequence with the pillar over the air tunnel being mined with a pillar hole as the last step in the recovery before mining the small stump pillars remaining along the slant chutes.

This plan eliminates the driving of 1080 ft of coal gangways and return airways for each 360-ft section, also the driving of four 30-ft breasts with a resultant saving in labor and materials. Better recovery should be obtained and the completion of each section should be speeded.

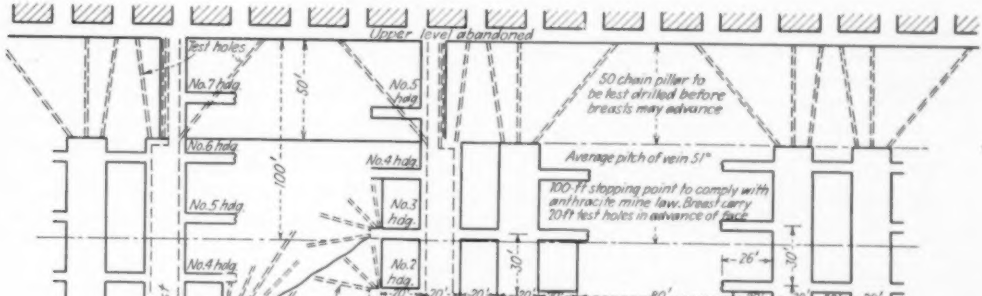
Recovery of Coal Below Rock Holes

In mining where the development is made with rock gangways underneath and parallel to the seam, and rock holes are driven from the gangway to the vein on 25 to 45°, considerable coal is left between the gangway level and the end of the rock hole. The amount of coal left standing depends upon the thickness of the vein, the pitch of the vein and the pitch on which the back chute was driven from the top of the rock hole to the top slate.

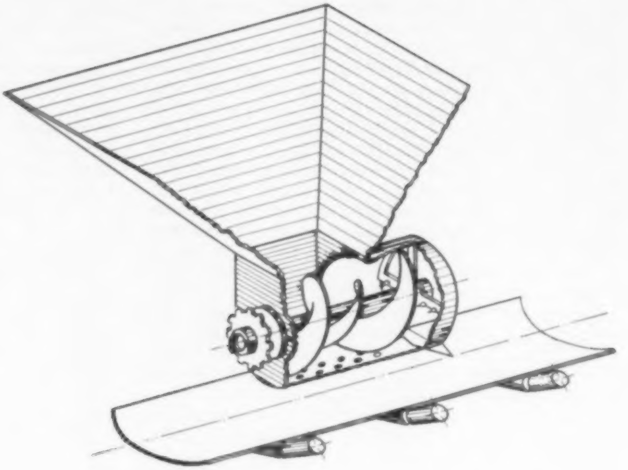
The coal is usually left standing to be recovered at some future time from a lower level. However, where the vein averages 20 or more ft in thickness, it can be recovered economically by driving horizontal holes to the vein from the rock gangway. A plan to recover this coal has been developed for one of the collieries in the region and will be started as soon as delivery of conveyor equipment is made. Horizontal holes or short conveyor gangways 6x10 ft in the clear are to be driven at right angles from the rock gangway to the top slate of the vein.

The conveyor gangways will be spaced on 40-ft centers which will leave a pillar of 30 ft to be mined between each opening or 15 ft of pillar along each rib. A small size chute will then be driven along the top slate of the vein until it connects with the back chute that was driven from the rock hole. This will give an open end for both ventilation and for firing of the pillar.

After the ventilation is established, three 1¼-in. holes are to be drilled in the coal to within 5 or 10 ft of the gob and fired with permissible explosives; one hole directly over the top of the conveyor gangway and one over each rib. The drilling and firing of additional holes along the conveyorway will be determined by the nature and quality of the coal. Ventilation when advancing and retreating is to be supplied by small fans installed near the rock gangway. The coal is to be loaded by shaking conveyors equipped with duckbills and telescopic troughs and discharged into an electric driven chain conveyor for loading into mine cars. This plan should eliminate considerable relief timbering and give good results as to recovery and performance.



Modified breast and pillar method now under consideration at Locust Gap colliery of Philadelphia & Reading.



Screw Crusher Solves Problem for Freeport

by A. A. Gustafson

The Freeport Sulphur Co. built a portable crusher in 1944 that solved a problem that none of the crusher manufacturing companies contacted were able to do. Two of these crushers have now been in service for 6 years, giving satisfactory service in handling a million tons per year, and having a capacity of over 2000 long tons per 8-hr shift. These machines reduce ton size blocks of sulphur to 12 in. and also feed the conveyor system at the rate of 350 long tons per hr per machine.

Mr. Gustafson is a mining engineer for the Freeport Sulphur Co., Grande Ecaille, La., and is a member of AIME.

This portable screw crusher undoubtedly has an application in other mines where it is desirable to reduce run of mine ore to conveyor size. Potash, coal, salt, clay, shale, and some iron ores appear to be materials which could be crushed by machines built on this new crushing principle. Some mines have hesitated going to conveyor haulage due to not having solved the crushing problem.

Before describing the crusher it will be of interest to describe the Freeport Sulphur loading operations at Grande Ecaille, 50 miles south of New Orleans, where the crushers are used. Liquid sulphur is pumped to two bin sites, supported on 60 and 75 piling, where it is cooled. Bins are 200 ft wide, 700 ft long, and 20 ft high. The main conveyor is located between the two bins. On each side of the main conveyor, a traveling cross conveyor, supported on three 60-ft trusses mounted on wheels, is moved across the 200 ft width of the bins as loading progresses. Besides supporting the conveyor installation these trusses also support a portable 5-ton capacity hopper mounted over the crusher-feeder. The hopper receives feed from a 2½-cu yd bucket, and moves along the cross conveyor on rails as the shovel digs across the sulphur bin.

From 1934 to 1944 lumps were caught on a grid with 12x18 in. openings. Two men on each shift broke the lumps to conveyor size with picks, but could not keep up with the 1¾-cu yd shovel. The hopper had a capacity of 5 tons of sulphur and was equipped with a pan feeder for feeding the conveyor. This was a costly and dangerous operation which suggested the use of a crusher, or heavier blasting to reduce the size of the lumps. Since we operated on marshy, filled ground, supported by piling, and it was necessary to use timber mats under the shovels, a

greater use of dynamite was not practical, or economical.

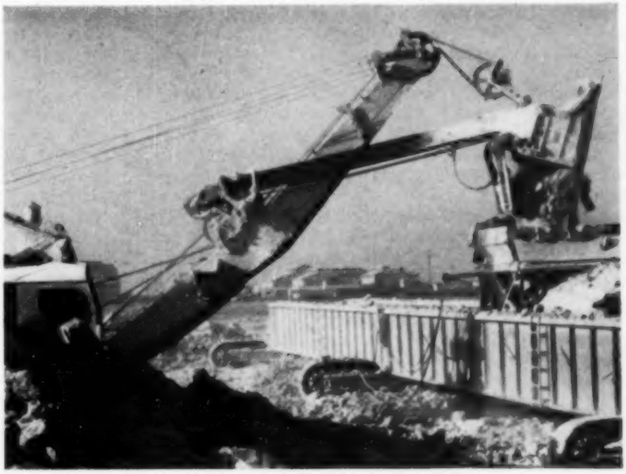
Consequently crushing was the answer, and most of the crusher manufacturers were contacted for a solution. The majority shied away from a 350 ton per hr portable crusher, but those who tried to work out something came up with more weight than the conveyor trusses would stand, and there was no way of strengthening the trusses at a reasonable cost. Coal crushers with their toothed rolls came the closest to meeting needs as far as weight went. However, when the manufacturer advised that coal crushers could not be choke fed with 1¾ tons at a time, and that a hopper, feeder, and screen would have to be added, the proposition had to be dropped.

One day in 1943, it was noticed that sulphur samples were being crushed by a meat grinder. This observation gave birth to an idea that resulted in the Freeport crusher which has been so successful in sulphur operations. A small experimental crusher was built and tried. Numerous changes were made as there was no information for engineering design.

The Freeport crusher consists of a screw operating in a trough welded to the bottom of a hopper. The screw also acts as a feeder to the conveyor belt, thus eliminating the usual pan feeder. The lumps that are too large to settle into the trough where the screw operates are broken off at the bottom by the screw ribbon until they settle down on the screw shaft. Then they are carried forward to the front edge of the hopper, where teeth attached to the sides of the frame split the chunks as the ribbon of the screw forces the chunks against the teeth. The isometric sketch shows the operation of the screw. The bottom of the trough was a solid plate originally, but due to accumulated moisture causing corrosion, holes were cut in the bottom to allow the water to get out. Some of the fine material now falls thru these holes on to the belt, and forms a cushion for lumps. The bottom of the trough can be perforated to screen out any amount of fines desired, and these openings together with inside stiffners on the bottom of the trough can aid in crushing. The crusher is designed to take feed 3 to 4 ft in diam and reduce it to chunks 10 to 12-in. size. The fine material is not reduced in size but fed to the conveyor belt by the screw along with the crushed chunks. The shovel dumps into a hopper over the trough which has a capacity of 5 long tons. No trouble is experienced in choke feeding the screw with this amount of material.

The shaft of the screw is 6 in. in diam. Two ribbons of boiler plate are welded to the shaft. The large ribbon is of 1¼-in. plate and bent to make a diameter of 30 in. The small ribbon is 18 in. in diam and made of ¾-in. plate. The ribbons extend along 4 ft of the shaft. The screw turns at 15 rpm, is driven by a 30 hp 1200 rpm motor, equipped with a 20-in. fly wheel, and is direct connected to a 32.5 : 1 reducing gear which is connected by sprocket and chain drive to the shaft. The trough in which the screw operates is made of ¾-in. plate. Originally only the larger ribbon was used and the first crusher cost about $3000 to build. Maintenance costs have been negligible. The crusher including hopper, drive mechanism, and supporting structure for traveling over conveyor belt weighs 22,000 lb.

This crusher can be built in larger or smaller sizes than the machine described above, and also with a variation in screw pitch. For some materials it may also be desirable to use two or more screws side by side. The rpm can also be varied to obtain the best results on various materials, and greatest efficiency in regard to horse power weight, and size of feed and crushed product.



Traveling cross conveyor moving across the sulphur bin as loading progresses. Five-ton capacity hopper is fed by 2½-cu-yd shovel, and hopper moves on rails as shovel digs across the sulphur bin.

Blast going off at the Bagdad open-pit

Drilling and Blasting at Bagdad Copper

by Olaf Hondrum

Mr. Hondrum is mine superintendent, Bagdad Copper Corp., Bagdad, Ariz.

CHURN drilling equipment at Bagdad consists of two Bucyrus Erie 27-T model drills and one 22-T drill with gasoline engines. The drilling tools weigh approximately 1600 lb. The holes are drilled with 7-in. bits which are heated in an oil burning furnace and sharpened in an electrically powered bit sharpener. A service truck brings the bits to the drills and takes the dull bits back to the shop. The bits weigh around 250 lb. when new and are handled by drill crews without any mechanical handling device.

When the management decided to start open-pit operations in 1945 the ruggedness of the terrain was one factor that led to the purchase of the small 22-T model drill. The orebody lies on both sides of Copper Creek Canyon. The sides of the canyon are steep and are cut by numerous narrow branch gullies. The drill had to travel over roads that were narrow and steep. It proved adequate for drilling the capping formation and is still in use.

The orebody at Bagdad is a chalcocite-enriched zone in a gray colored monzonite porphyry. Generally speaking, it is a fairly hard formation. Drilling speed averages 40 ft per shift with two bit changes per hole. The capping has been thoroughly oxidized and is softer than the ore, except where it is cut by silicious veins or ribs. Sixty ft per shift represents average drilling speed in the capping. Occasionally a bit will last through a shift but the average footage per bit is less.

Benches vary in height from 45 to 50 ft. Churn drill holes are spaced 15 to 18 ft apart in the capping and 12 ft in the ore, and are drilled 5 ft below grade. In starting a hole a short piece of casing is used. Upon completion of the hole it is pulled up and used in the next hole. Only very occasionally is it necessary to leave the casing until blasting. The outer row of holes is drilled 10 to 12 ft from the edge. Two or three rows of holes with a total of 30 to 35 holes represent an average blast.

Bag powder is used, except for primers, which are sticks of gelatine dynamite. Two grades of powder are used, one of 60 pct strength and the other 20 pct. They are used in about equal amounts. From past experience the powder foreman judges the amount to be used in each hole, taking into account the estimated burden, hardness and type of formation. A typical charge in a 55-ft hole in ore is 7 cases of powder, rising about 21 ft in the hole and leaving 34 ft for stemming. This results in satisfactory fragmentation. In capping material the powder charge is less, with a stemming column of about 40 ft except for holes drilled in more than average silicious material. In this case more powder is used, or the hole may be deck loaded in order to bring the powder charge closer to the top.

In the extreme east end of the orebody the capping consists of a tight conglomerate which is fairly easy to drill but hard to break. Deck loading of holes is standard procedure in this formation. Placing all the powder in the bottom resulted in the top half of the bench remaining unbroken. Better fragmentation is obtained by using a slow speed powder and bringing the powder charge within 20 ft of the top by deck loading.

To set off the charge, double strands of Primacord from the holes are connected to a double trunk line. A dynamite cap with fuse is attached to each end of the trunk line. This method is wasteful of Primacord but inasmuch as it helps to keep down misfires it has been adopted as standard.

On The Face of it....

ON the face of it, the American Mining Congress meeting was a big success, as witness the smiles on the AIME members, pictured here, who attended. The grins were captured at Booth 232, where caricaturist Lenn Redman held forth for the Independent Pneumatic Tool Co., sketching 114 subjects in one week.

Some of the grins were occasioned by the news from Simon D. Strauss, vice-president of AS&R, that the outlook for nonferrous metals is bright, in view of the extraordinary consumption by both the civilian and military populace in recent months. Then, some were happy to hear North Carolina's Representative Carl T. Durham advocate the support of marginal mines and the stockpiling of metal from foreign sources. He also called for centralization of the stockpiling program in one central agency.

A lot of the smiles disappeared when Roy Hatch, assistant general manager of Kennecott's Utah Copper Div. estimated that 23 pct is automatically added to pre-set wage rates by such expenses as overtime, unemployment compensation, insurance, housing, recreational facilities, etc. He saw a threat to the nonferrous industry in further union attempts to increase these benefits, the cost of which would not be transmitted directly to the consuming public.

Gayety, however, prevailed at the Mining Congress cocktail party on Sunday night, which was jammed like the first cage going off shift. Then, on Monday night, 'way down at the bottom of Bingham Canyon, engineers and their wives stuffed themselves with spaghetti, beer, and highballs, and partook of a bit of gambling, a bit of dancing, and a look at a fine variety show. A steak dinner at Salt Lake's Lagoon amusement park provided the buildup for Thursday night's social crescendo, the very memorable banquet. Turn the page for more convention news.

American Mining Congress President Howard I. Young happily surveyed the biggest metal mining convention and exposition yet held.


At top, from left: Ben H. Cody, Phelps Dodge Corp, Morenci, Ariz.; E. S. McGlone Anaconda Copper, Butte, Mont.; M.D. Harbaugh, Lake Superior Iron Ore Ass'n, Cleveland; John W. Chandler, Eagle-Picher, Miami, Okla.; **Below:** John J. Curzon, Howe Sound Co., Holden, Wash.; and Wesley P. Goss, Magma Copper Co., Superior, Ariz.










In the center, Guy N. Bjorge, Homestake Mining Co., Lead, So. Dak.; and, with the cigar, Neil O'Donnell, Idaho-Maryland Mines, Grass Valley, Calif. **Above,** and reading down to the right: J. F. Buchanan, Magma Copper Co., Superior, Ariz.; H. J. Rahilly, Anaconda Copper, Butte, Mont.; C. N. Kravig, Homestake Mining Co.; P. D. I. Honeyman, Inspiration Copper Co., Inspiration, Ariz.; C. E. Weed, Anaconda Copper, New York; and H. C. Weed, Inspiration Copper Co.

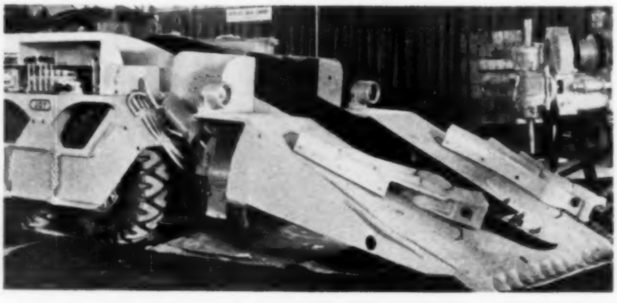

Record Crowd at AMC Meeting

WARMED by the Utah sunshine and cheered by the lavish hospitality of suppliers, over 2000 mine operators thronged the Fair Grounds in Salt Lake City, Aug. 28 to 31, to view the equipment displays and sit in on general and operating sessions. Total registered attendance at the convention and exposition reached 5588 by closing Thursday night, of which 1838 were manufacturers of mining equipment, 2482 were operators, 1063 were lady guests, and 205 were visitors. The record attendance was matched by a record number of equipment displays which numbered more than a hundred. M. L. McCormack, chairman of the AMC Manufacturer's Div., and his associates certainly did a splendid job in arranging the exhibition continuous hospitality. One fine example: the Welcoming Luncheon, pictured above.

Although each equipment display was sufficiently interesting in itself to draw spectators, various gimmicks were used in friendly competition by the exhibitors to outdo each other in drawing visitors. Opportunities to win such items as a typewriter, a hunting rifle, Scotch and similar items were numerous. All one had to do was to guess the weight of an unidentified mineral sample or the number of blasting caps in a fish bowl, etc., to carry away one of these items.

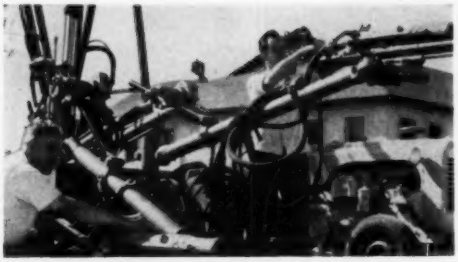
Although Salt Lake is ostensibly dry, alcoholic refreshments were available at any hour of the day or night from Sunday through Thursday thanks to the manufacturers' suites. In addition, the several special cocktail parties held were extremely elegant. Entertainment ranged from a variety show at the Eimco party to a string quartet.

At the Colorado Fuel & Iron Corp. exhibit Otto Herres, vice-chairman, exposition committee, Will I. Powell, W. Lunsford Long of the Haile Gold Mines, and the Hon. Graham A. Barden, Congressman from North Carolina, view a model CF&I ball mill in operation. The mill had glass back and front to enable visitors to view its operation.

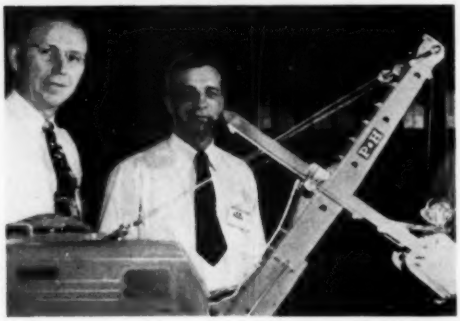


One of the features of the Joy Manufacturing Co. exhibit was this Joy 17-HR continuous type loader. Joy also showed a drillmobile, stopers, sluchers, hoists, rock and core bits and a variety of other products for the mining industry.

Mack Trucks, Inc., was well represented by this model LSRW 30-ton 6-wheel off the highway dump truck, exhibited outside the main exposition hall.

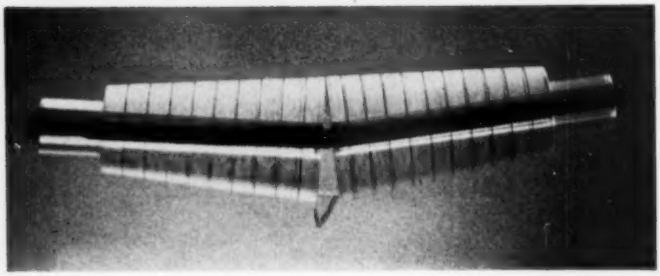


Left: E. G. Simmons and Tom Stone were on hand to show visitors this mine jumbo manufactured by the Cleveland Division of the Le Roi Co. Below, left: J. W. Hill and E. M. Paris, both of the U. S. Vanadium Corp., enjoyed the working model of a P&H magnetorque shovel shown by the Harnischfeger Corp. Below: Big 10-ton model UD rear-dump truck keynoted the Euclid exhibit, outside the exposition hall.



Out in the noonday Salt Lake sun, another big exhibit captured visitors' attention. It's the Caterpillar Tractor Co.'s diesel wheel-type tractor with an Athey wagon.

Self-Centering Conveyor Roll Expected to Revolutionize Materials Handling



A new and startling advance in the field of materials handling has come from the Carnegie-Illinois Steel Corp., with the first announcement of a self-centering roll which completely eliminates the necessity for sideguides to keep materials on a conveyor. The "almost unbelievable performance" of these rolls has been demonstrated even on circular conveyor tables. Objects being rotated on the table did not deviate from their lines of travel even when the table was tilted from the horizontal.

Self-centering rolls are now being manufactured at Carnegie-Illinois' Johnstown Works, and they are now in use at the McDonald Works in Youngstown as well as at Tennessee Coal & Iron at Fairfield, Ala. Two stainless steel conveyor belts are being contemplated for test at the Lynch, Ky., mine of the H. C. Frick Coke Co., and at the Bituminous Coal Research Institute's installation in Virginia.

The Lorig roll, invented by E. T. Lorig, head of Carnegie-Illinois' senior engineering staff, is a slightly crowned roll cut transversely at the center. The two halves are fixed to rotate as a unit, with the working surfaces approximately horizontal and the axes at an angle. Thus the lines of force in both halves twist evenly toward the center in the direction of movement, conferring a self-centering action on materials passing over the rolls. The planar action of the rolls exerts a greater force than gravity, as has been shown by operating pulleys vertically and attempting unsuccessfully to pull a metal belt down against the lower pulley housings.

The device has been tested for all manner of applications in the steel industry, and there is now said to be "no question" that applications for the roll will be found in other industries. **Circle No. 1**

Bore Hole Loader Eliminates Tamping Poles

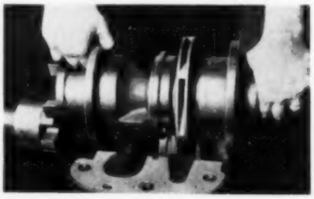
One of the sensations at the AMC metal mining exposition in Salt Lake City was the Atlas Bore Hole Loader, which makes use of a long metal-cored rubber hose to perform loading and tamping operations in holes up to 80 ft long. Several cartridges can be pushed at one time to the back of a hole, and firmly tamped by retracting and advancing the plastic-tipped hose. Tamping poles are eliminated and manpower requirements sharply reduced. An ordinary compressor, such as that used for drilling, operates the loader, which comes in two sizes. The smaller size, (illustrated) is hand-held, has a hose diam of ⅞ in., and can load holes up to 80 ft long. The larger model is equipped with a tripod and enough hose for a 70-ft hole. **Circle No. 2**

New 25-cu yd Truck

Featuring a heavy-duty body with a struck capacity of 25 cu yd, and an 8-cylinder diesel engine, this new off-highway dump truck has just been announced by Sterling Motors Corp. A double-acting hoist provides a 65° dumping angle for easy unloading. A 4-speed main transmission and 3-speed auxiliary transmission provide 12 well-graduated forward and three reverse speeds. The cab is of the one-man type, offset to the left. The truck was designed primarily for use in coal stripping, open-pit mining and quarrying. **Circle No. 3**

Pump Repair Simplified

A new pump in which all working parts are contained in one easily removed rotor has been designed by the De Laval Steam Turbine Co. In addition, simple mechanical seals replace the stuffing box, and the pre-lubricated bearings in the pump are lubricated for life, thus making it even more "maintenance free." When maintenance is necessary, top cover and end plate studs are removed, and the rotor assembly is taken out and replaced by a new one. **Circle No. 4**

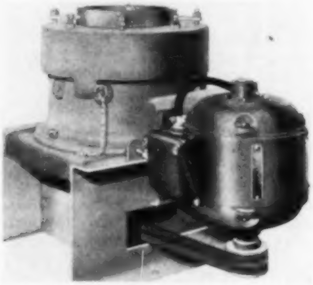


New Hex-Shank Auger Bits

Tungsten-carbide tipped auger drill bits having hex-shanks are a new product of the Firth Sterling Steel & Carbide Corp. The line was developed expressly for drilling holes for Armstrong, Airdox or Cardox shooting. Bit diameter sizes range from 2¼ in. with 13/16-in. hex-shank up to 3¼ in. with 1⅜-in. hex-shank. Several of the diameters are furnished with a selection of shank sizes. **Circle No. 5**

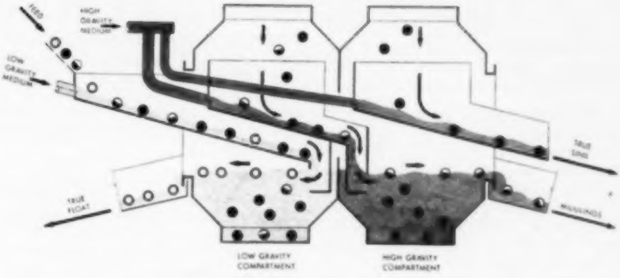
Reduction Crusher for Labs, Pilot Plants

A laboratory or pilot plant reduction crusher designed to reduce ½-in. feed to as fine a product as 10 mesh (single pass) is offered by Mine & Smelter Supply Co. It is



said to provide an operation and product comparable to that obtained by commercial crushers, and replaces bulky equipment such as rolls and the coffee mill. Product can be used as feed for ball or rod mills, laboratory pulverizers, and tables, sink-float, or spiral concentration. Manufactured in two sizes, 6 and 10-in. **Circle No. 7**

Two-Compartment Drum Separator Announced



WEMCO TWO COMPARTMENT DRUM SEPARATOR

A completely new drum separator, which reduces the complexity of two-stage separation by using a partitioned drum, and which is expected to have wide application in the cleaning of low grade coal and in the beneficiating of iron ores, has been announced by the Western Machinery Co. The WEMCO process uses a partitioned drum with a lighter media in one section and a heavier media in the other, as illustrated. Material moves from one section to the other, eliminating screens, conveyors and duckwork necessary when separate vessels are used. Three conveyor systems leave the drum, carrying a true mineral product, a true waste product, and a middling product. Three substantial savings are claimed: fewer items of equipment, lower construction and installation costs, and greatly reduced floor space. **Circle No. 8**

Photoelectric Smoke Indicator

The Combustion Control Corp.'s smoke indicators are designed to continuously indicate the density of smoke passing through a stack and to signal when it passes a predetermined value. This serves to keep smoke conditions within the limits of local ordinances and maintains a check on combustion efficiency. Light source, photoelectric scanner, control and indicator are combined in one housing. Red and green lights indicate smoke conditions. Supplied with or without 30-watt voltage regulator. **Circle No. 6**

Manufacturers' Bulletins

For your convenience, a listing of booklets and other material currently being offered by the manufacturers. To obtain this information, merely circle the desired number on the coupon, and return it to MINING ENGINEERING.

9) AERIAL PHOTOGRAPHY: Many mining companies are making full use of aerial photography and airborne magnetometer surveys for exploration and prospecting. Map facts of the roughest kind of terrain are collected in a matter of days compared to the weeks and months needed to explore the area with ground parties. *Aero Service Corp.* has issued a 24-page booklet which describes the many advantages of using this service for engineering and geologic study.

10) SELF-PRIMING PUMPS: Pumps of a new design that eliminates valves and yet gives efficiency comparable to standard centrifugal pumps are described in bulletin 636.1 available from *Goulds Pumps, Inc.* These self-priming pumps are made in sizes ranging from ¼ hp to 5 hp with both open and closed impellers. Cross-sectional illustrations listing the parts used in the manufacture of the pump are shown.

11) GYROCENTRIC SCREENS: A new brochure on the use of screens for wet and dry separation of chemicals, crushed rock, abrasives, refractories, dewatering and liquid recovery processes has been published by the *Patterson Foundry & Machine Co.* Installation photographs, screen specifications, and detailed information about 22 standard screens are given.

12) INDUSTRIAL RESEARCH: "Research Facilities Without Capital Investment," is the title of a new booklet offered by *Foster D. Snell, Inc.*, dealing with the position of the consultant and research work in today's industrial research. The booklet briefly outlines the relation of equipment and personnel of the consultant and the results to be expected on various types of problems.

13) METER ADAPTER: An adapter is now available for installing Flowrator meters in horizontal process piping. This eliminates the necessity for a vertical rise in a line where headroom is at a premium. Eye-level flow indication for accurate instrument reading is now made possible by this adapter, where necessity of vertical runs made it impossible before. Catalog section 25 may be obtained from *Fisher & Porter Co.*

14) QUARTZ SPECTROGRAPH: A newly revised bulletin available from

Gaertner Scientific Corp. describes and illustrates all of the principal features of this precision research instrument. Dispersion curves for the interchangeable quartz and glass optics are included with specifications, optical diagram, and list of recommended accessories.

15) PROVING RINGS: Direct reading proving rings which are used in calibrating the loads of various types of testing machines and presses, including Brinell and Universal testing machines are described and illustrated in a brochure published by *Steel City Testing Machines, Inc.* The file-size data sheet gives full information on sizes, capacities, weights, and dimensions of the instruments.

16) VARIDRIVE MOTORS: Several of the features of these motors include a new finger touch control handle, smaller dimensions, and optional positioning of the control dial. The seven modifications of the design including three-phase and single phase, combination geared drives and types with flanged bracket for direct connection to the driven machine are illustrated in bulletin 1601 available from *U. S. Electrical Motors, Inc.*

17) FAN VENTILATOR: Especially suitable for situations where powered duct exhaust ventilators operating at very low noise levels are desirable, a new centrifugal type fan with backwardly curved blades, mounted within a weather-proof chamber, is described in pamphlet 341 available from *Swartwout Co.*

18) SAFETY HAT: A new illustrated circular available from *E. C. Bullard Co.* contains complete information on a new ribbed design metal safety hat. The inner hat assembly is designed to absorb shock with its full-floating, six-point suspension hammock. The entire assembly can be replaced without tools with slide action wedge type fasteners.

19) WIRE ROPE: Condensed information on improved plow steel wire ropes is contained in bulletin 50-25 which may be obtained from *Macwhyte Co.* All sizes and construction classifications are combined in one large table.

20) DIESELS: A new line of heavy-duty engines, power units and generator sets capable of burning either natural gas or diesel fuel is described in catalog 107 issued by *Murphy Diesel Co.* These heavy-duty gas engines utilize the economical advantages of high compression combustion. Three models are available ranging from 135 to 180 hp.

21) CONVEYORS: Two new pamphlets have been issued by the Conveyor Co. covering new information on troughing idlers and return idlers of improved design which eliminate high-speed rattle. Applications of these improved idlers vary from operations in mines and smelters to pits and quarries. The features of the new snug design and its application to permanently lubricated ball-bearing type idlers and roller bearing idlers are included.

22) CONCENTRATOR: The Weinig concentrator described in bulletin 50 available from *Colorado Iron Works Co.* has been developed to bridge the gap between heavy density separations and flotation or other methods of treating extremely fine sizes of liberated mineral and gangue. The machine consists of two units, a dewatering tank and spiral mechanism for removing the concentrate, or heavy product, and the concentrator itself in which the separation of sink and float products is made. It will make efficient separations of materials in a size range of ¼ in. to between 28 mesh and 35 mesh with a difference in specific gravity as low as 0.20. Some fine materials that have responded to the treatment are iron ore, anthracite, dump ores containing complex sulfides and precious metals and bituminous coal.

23) DIESELS: A 32-page booklet available from *General Motors Corp.* tells the story of modern diesel power. Beginning with the first diesel built and progressing through the years, the booklet has many illustrations. The complete operation of the diesel is illustrated with cross-sectional diagrams of the four-stroke cycle, two-stroke cycle, the fuel injection system and other pertinent details.

24) WHEELS AND HITCHINGS: The advantages of using Naco steel wheels and hitchings are described in circular 1746 issued by *National Malleable & Steel Castings Co.* Tests conducted on these wheels show the first permanent set did not occur until above 75,000 lb. They can be made considerably lighter than the chilled cast iron wheel, thus reducing the tar weight per car.

25) HOSE LINES: Single cotton braid, single rayon braid, and double wire braid hose are just a few of the different types of industrial hoses available from *Aeroquip Corp.* described in booklet. Fittings and assemblies to be used are also illustrated and the pipe thread, SAE thread and approximate weights are listed.

26) SHAFTS AND PINIONS: One of the few applications of *Pittsburgh Gear & Machine Co.* taper serrated shafts and pinions is on mine locomotive armature. Cutting away metal on shaft and under pinion tooth on old keys and keyways weakens both. Using this new way all the original metal and strength of shaft and pinion are retained.

27) AIR FEED SINKERS: These light drills combine the drilling speed of big power feed drifters and the portability of hand hold sinkers with the air consumption of 45-lb and 80-lb jackhammers. Described in pamphlet RD6 available from *Le Roi Co.* these drills make every phase of drilling more efficient. Reverse air feed sinkers are up to 70 lb lighter than power feed drifters of like capacities.

28) LUBRICATION: The fundamentals of correct oil lubrication are detailed in an 8-page color booklet available from *Socony-Vacuum Oil Co., Inc.* The basic principles of boundary and flood lubrication are illustrated by diagrams.

Mining Engineering — October
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Roof Studies and Mine Structure Stress Analysis, Bureau of Mines Oil-Shale Mine, Rifle, Colo.

by E. M. Sipprelle and H. L. Teichman

ENACTMENT of Public Law 290 by the 78th Congress authorized the U. S. Department of the Interior, Bureau of Mines, to conduct an experimental program to develop the technology for obtaining oil from oil shale. In adopting and later extending this legislation, the Congress recognized the impending necessity of supplementing ground petroleum reserves with synthetic fuels. Under the provisions of this legislation, the Bureau of Mines, among other things, was charged with the responsibility of developing mining techniques, methods, and equipment for mining the oil shales of the Green River formation of Colorado, Utah, and Wyoming.

The oil shales of western Colorado are apparently richer, more accessible, and more amenable to exploitation than elsewhere in the Rocky Mountain region. The site chosen for the Bureau's Experimental mine is about 10 miles west of Rifle in northwestern Colorado. It is within a 1000-sq-mile area from which, it has been estimated, 300 billion barrels of shale oil could be produced from a 500-ft measure near the top of the formation. One hundred billion barrels of this amount could be produced from the Mahogany ledge, a 60 to 100-ft section near the bottom of the 500-ft measure. This ledge is considered to have economic importance at present.

The Green River formation was laid down as sediment in the bottom of vast, shallow inland lakes during Eocene time. The deposit is flat-lying, and there are no faults, fissures, or local rolls. Oil shale is actually a strong, tough magnesium marlstone, which will stand unsupported over relatively wide spans. These and other natural physical characteristics favor mechanized, low-cost mining, which is essential for establishment of an oil-shale industry.

It was realized from the outset that an extensive research program would be necessary to develop mining methods, equipment, and techniques for a mechanized, low-cost operation. The program was designed to include research into all the productive phases of mining, such as drilling, blasting, loading, transportation, and maintenance of the mine structure. The methods, equipment, and techniques developed as a result of this research have established a production of 116 tons per man-shift total labor at a direct cost of $0.292 per ton.

Another important phase of the research program that has received little publicity because of its theoretical nature is study of the roofstone behavior and determination of mine structure stresses. This paper purposes to discuss this phase of the research program.

Preliminary studies of the physical properties of the Green River oil-shale formation were made in the Barodyramics Laboratory at Columbia University during the latter part of 1945 and the early part of 1946.* The purpose of these studies was to determine the maximum size of unsupported underground openings that would be commensurate with safety and still permit the use of large, efficient mining equipment. Also to be determined were the pillar support to extraction ratio and the shape, size, and spacing of supporting pillars. Selected samples of possible roofstones near the top of the Mahogany ledge, as well as representative samples of different rock types found within the ledge, were obtained from the Bureau's oil-shale mine for these studies. The maximum safe unsupported roof span calculated from this work was 200 ft. Using a safety factor of four, it was theoretically determined that openings 60 ft wide could be advanced under a roofstone at the top of the Mahogany ledge. To support the overburden, 60-ft-sq pillars would be left in a checkerboard pattern.

From visual observations made of core samples through the selected roofstone at the oil-shale mine, it was determined that the roofstone was actually a plate 6 to 8 ft thick.

Because the calculations were theoretical and allowance had to be made for unknown cracks and fractures in the formation, openings 50 ft wide and pillars 60 ft sq were originally contemplated in the Bureau's Experimental mine. This would be the minimum allowable width that would permit use of large underground mining equipment. For lower mining costs and greater efficiency larger openings were desirable.

Different but analogous approaches were made to the problem at the Bureau of Mines Applied Physics

* F. D. Wright and P. H. Bucky: Determination of Room and Pillar Dimensions for the Oil-Shale Mine at Rifle, Colo. Trans. AIME, 181, 352; Min. Tech., Nov. 1948, TP 2489.

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AIME New York Meeting, February 1950.

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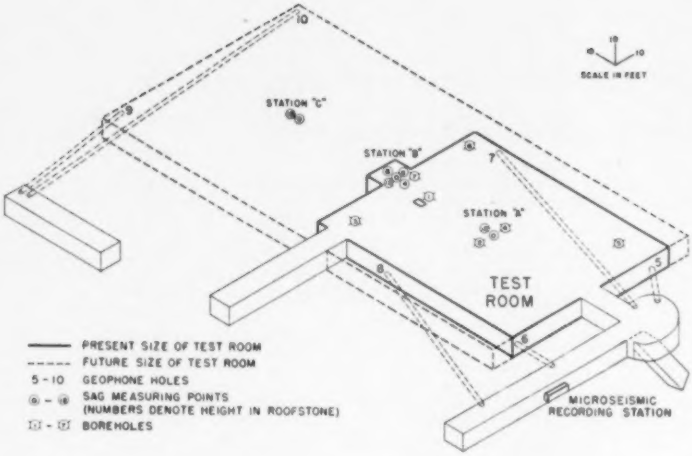

Fig. 1—Isometric drawing of test room showing present and proposed size and recording installations.

Laboratory, College Park, Md., and the Bureau of Mines Pittsburgh Station. Essentially the same results were obtained.

Seismic Investigations

To check the laboratory data and theory and to prove conclusively the maximum size at which underground openings could be mined safely, it was decided to excavate an experimental test room 50 ft wide by 100 ft long directly under the selected roofstone. In this test room it would be possible to study the various physical properties of oil shale as a formation in place and to demonstrate the practicability and use of scientific instruments in mining research under controlled conditions.

Excavation of the 50x100-ft test room was completed in December 1946. By February 1947 specialized equipment had been installed, and the first scientific studies were started. The procedure has been to widen the room in increments of 10 ft, while using the specialized equipment to make daily observations of the behavior of the roofstone. Each new width was observed for a period of time until it was definitely determined that no excessive stresses were developing. The test room was widened from 50 to 60 ft in June 1947 to 70 ft in August 1947 and to 80 ft in November 1948 (fig. 1). During this period it was observed that small hairline cracks developed in the roofstone, and air slacking of a ½-in. layer on the roofstone occurred to a minor degree. Aside from the hairline cracks and air slacking, no other failures have occurred. During the entire period of widening from 50 to 80 ft, while under constant observation, no indications of excessive stress have been observed.

Table I. Typical Noise Levels Compared to Sag Rates

Noises per Hour	Approximate Avg Sag, In.	Remarks
0 to 5	0.0001	Average noise level of test room with no mining activity. Considered very quiet.
20 to 60	0.001	Considered noisy with some possibility of approaching the elastic limit. This is approximate rate immediately after widening is completed. The rate dropped to a normal level within a few days.
100 to 1,000+	0.01	During the widening of test room. Would be extremely dangerous if not due to blasting. This noise level decreased rapidly and several days after blasting the rate dropped to a normal level.

One generally accepted method of predicting imminent failure in underground mine openings is by listening to the ground "work," that is, make audible popping and cracking noises. These noises are caused by failure of rock crystals and actual cracking of the rock formation when stressed beyond the breaking strength. The intensity and frequency of these noises offer a semiquantitative method of predicting excessive stresses in underground mine structures. It was reasoned that if these noises could be detected while still in the subaudible range, the maximum safe roof span in the oil-shale formation could be determined.

Seismic equipment (fig. 2) was adapted to detect and record these subaudible noises caused from cracking or rearrangement of minute rock crystals subjected to pressure. The equipment consists of geophones, amplifiers, and recorders. The geophone is a metallic cylinder, 1¼ in. in diam and 8 in. in length, which houses a small Rochelle salt crystal. Subaudible rock noises are picked up and transformed into small electric currents by the salt crystal. These electric currents are amplified and recorded as pulses on the microseismic recorder tape. Four such geophones were installed in the roofstone of the test room near the four corners where maximum stress is most likely to occur. Each geophone makes a separate record. Minute rock noises can be detected by a geophone in solid ground from a radius of 50 to 70 ft. The recording from a geophone closest to a minor local disturbance will show greater frequency and intensity than one farther away. Disturbances of a general nature would be recorded nearly equal in frequency and intensity from all geophones. Records of the rock noises are made each day and the results tabulated.

From a study of these data it has been possible to establish a definite noise level (number of noises per hour) pattern that gives a qualitative indication of the stress conditions in the test-room roofstone. The average normal noise level is less than five noises per hour. This average has been established from the seismic data recorded over a period of

several months between each widening of the test room. During these periods no mining activity has been conducted in the vicinity of the test room. Immediately after blasting in the test room or its vicinity and for a short period after each widening, the noise level has been as high as several hundred noises per hour. In all cases, so far, the noise level has dropped to the normal level within a few days. As long as this condition exists, the stresses are well within the elastic limits of the rock. If the noise level should suddenly start increasing or if it should fail to return to the normal level after blasting or widening, it could be assumed that the elastic limit of the rock was being approached.

Subsidence

It is an established fact that materials deform when subjected to stresses over long periods of time. This deformation, when smaller than a certain rate, does not materially affect the strength of the material. Laboratory tests have shown that shale has similar plastic properties. From these facts it was reasoned that the test-room roofstone would subside or sag to some extent because of its own weight and the pressure of overburden above it. Early in the test-room studies it was decided to measure this subsidence as accurately as possible and to correlate the subsidence rate with the seismic noise level and other stress data that could be obtained.

The original subsidence station was installed to measure the sag in the immediate roofstone at the center of the room. Stainless steel studs were cemented into the roof and floor at the center of the room. Periodic measurements are taken between these two points with an extensometer (fig. 3), an instrument made of stainless steel with an Ames dial on one end and a thermometer for making temperature corrections at the time of reading. Measurements taken with this instrument are accurate to 0.001 in. As the test room was widened, it became apparent that two main factors were affecting the sag. One was due to the width of the opening, in which case the sag was immediate and relatively large. The other was due to plastic flow causing a fairly constant but relatively small sag.

The greatest sag rate (0.13 in.) occurred between August 1 and August 25, 1947, when the room was widened from 60 to 70 ft. After widening, the rate

Fig. 2—Engineer taking readings at microseismic recording station.

(Photograph by U.S. Bureau of Mines)

Fig. 3—Engineer making sag measurements with extensometer at Station "A."

(Photograph by U.S. Bureau of Mines)

of sag decreased rapidly and stayed at a slow rate for several months, gradually decreasing with time and finally approaching zero. A graph of the sag from the time measurements were started to the end of 1949 is shown in fig. 4.

The magnitude of the immediate sag during widening from 50 to 60 ft and from 70 to 80 ft was less than from 60 to 70 ft. The reason for this is not definitely known. It could be caused by the longer time element and greater number of blasts during widening from 60 to 70 ft. Evidently blasting procedure has some effect on the sag. Exactly how much has not been determined.

The total sag measured from November 1948 to January 1950 (the period after widening from 70 to 80 ft) was only 0.10 in.

Roof Sag and Noise-Level Correlation: Correlations were made between roof sag and noise level. It was found that with each increase in sag rate a corresponding increase in noise level was observed.

Some typical noise levels compared to sag rates are shown in table I.

Apparent Thickness of Roofstone: From the subsidence data, the dimensions of the room, and the elastic properties of the roofstone, it was possible to calculate the apparent thickness of the roofstone by using the beam formula. Since there are several cracks running across the roofstone, using the beam formula instead of the plate formula will give results toward a safer value and will also conform more with the actual conditions.

From the beam theory

$$c^2 = \frac{p}{128E(d_2 - d_1)}(L_2^4 - L_1^4);$$

where c = ½ thickness of roof,
p = density of rock = 0.08 lb/in.^{3},
E = Young's modulus = 3x10^6 lb/in.^{2},
d_1 = deformation before increasing room width,
d_2 = deformation after increasing room width,
L_1 = room width before widening,
L_2 = room width after widening.

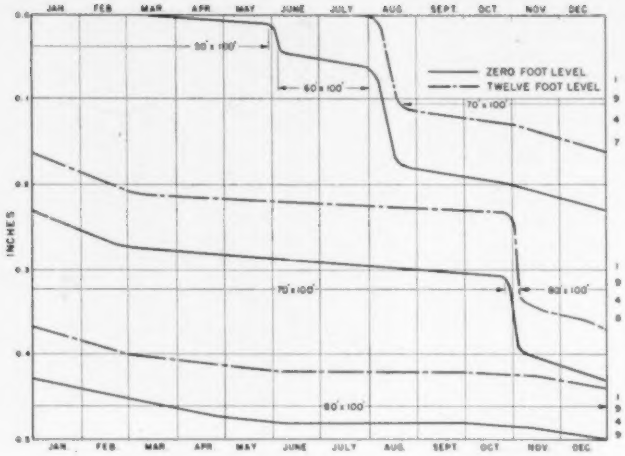

Fig. 4—Test room roof sag, Station "A."

For 50 to 60-ft widths,

$$c^2 = \frac{0.08}{128x3x10^6x0.03}(60' - 50')\ 12',$$

$$c = 31.08 \text{ in.} = 2.58 \text{ ft},$$
$$2_c = 5.16 \text{ ft}$$

For 60 to 70-ft widths,

$$c^2 = \frac{0.08}{128x3x10^6x0.12}(70' - 60')\ 12',$$

$$c = 10.68 \text{ in.} = 1.64 \text{ ft},$$
$$2_c = 3.28 \text{ ft}$$

For 70 to 80-ft widths,

$$c = \frac{0.08}{128x3x10^6x0.1}(80' - 70')\ 12',$$

$$c = 88 \text{ in.} = 7.33 \text{ ft},$$
$$2_c = 14.66 \text{ ft}$$

Average of all three determinations

$$\text{Avg} = (5.16 + 3.28 + 14.66)/3 = 7.7 \text{ ft}.$$

Differential Sag: To check this apparent thickness and locate any separation in the roofstone, differential roof sag measurements were started in July 1947. A special 12-ft extension stud was designed to fit into an AX diamond drill hole and to clamp into the roof 12 ft vertically up from the surface. Differential sag measurements were then obtained by subtracting the sag at the 12-ft level from the sag at the 0-ft level, thereby showing any separation that occurs between these two measuring points. Sag measurements 12 ft above the roofstone indicated that a maximum differential of 0.025 in. occurred during the middle of August 1947. This differential decreased with time, and at the end of 1949 the separation in the roofstone layers was only 0.002 in. This differential sag is only what has been measured. The separation is probably greater, since it is likely that separation to some extent occurred before differential measurements were started. This surmise is substantiated by other tests made in examining bore holes in the test room, which will be explained later in this report.

To supplement differential sag measurement, locate

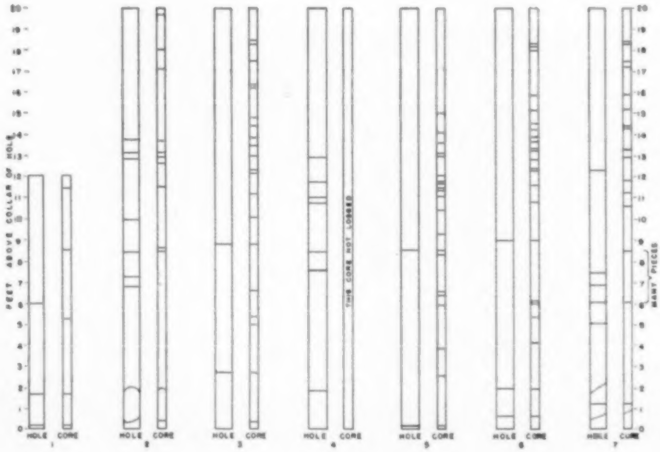

Fig. 5—Stratoscope and core log data from diamond-drill holes in test room roofstone showing location of cracks.

separations in the roofstone accurately, and gain other important information, it was desirable to examine the roofstone layers in the test room visually. One way of doing this is by using an instrument called a boroscope, with which vertical drill holes can be examined for cracks. An instrument of this sort, but called a stratoscope, was obtained from the Roof Control Section, Coal Mine Inspection Division, Bureau of Mines, College Park, Md. Seven vertical NX diamond drill holes averaging 21 ft in depth were examined, and photographs were obtained of cracks in the roofstone of the test room.

A crack 2 ft above the roofstone was observed in boreholes 1, 2, 4, and 6 and in a small raise near the center of the test room. A crack 8½ ft above the roofstone was observed in boreholes 2, 3, 4, 5, and 6. Another crack 13 ft above the roofstone was observed in boreholes 2 and 4. Other minor cracks were observed in some of the bore holes, but correlations with other holes could not be made. A summary of these data together with core logs from the same holes is shown in fig. 5.

Two drill holes 9 ft deep were examined in the roofstone of the Underground quarry. These were percussion drill holes, and the walls were rough, making it difficult to determine accurately the location of cracks, if any. There was some evidence of a small crack 2 ft from the roofstone in one of the holes. It is planned to repeat the examinations in boreholes in the Underground quarry roofstone at some future date.

Stress Measurements

In addition to the seismic and subsidence information, it was desirable to determine the absolute pressures in the mine structure. A method for obtaining these measurements was devised based on the theory of calculating the stress by the relief of strain.

The procedure followed is to select a site relatively free from blasting fractures. A surface about 4x5 in. is made flat and smooth by sanding. An electric strain gauge of the Baldwin SR4-type is bonded to the surface (fig. 6), and a zero reading is obtained with a strain indicator. Using a specially designed diamond saw and with the gauge in place, a prism section about 3x4 in. to a side is sawed out, containing the gauge (fig. 7). A reading of the strain relief is then obtained (fig. 8). In the laboratory, the moduli of elasticity are determined from the sawed-out specimen. From the moduli of elasticity and the strain relief, the original stresses in the specimen are calculated.

A number of strain measurements were made during 1949 in the oil-shale mine. Two types of measurements were made, one where the site tested was under compression and the other where the site was under tension. So far, all tension tests have proved unsuccessful, largely because tension was relieved by tiny cracks in the specimen. Compression tests, however, have yielded very encouraging results. Many of the measure pressures obtained came within 10 to 15 pct of theoretically calculated values.

For a 45° Rosette strain gauge where e_1, e_2, e_3 are the measured strains in the 0°, 45°, and 90° directions, respectively, the principal strains are given by:

$$e_p = A + B,$$

$$e_q = A - B,$$

Fig. 6—Engineer preparing to saw a strain specimen from the test room roofstone. Strain gauge may be seen bonded to the roofstone.

(Photograph by U.S. Bureau of Mines)

where

$$A = \frac{e_1 + e_3}{2},$$

$$B = \sqrt{\frac{2}{2}}\sqrt{(e_1 - e_2)^2 + (e_2 - e_3)^2}$$

The principal stresses are given by:

$$p = \frac{E\,(e_p + \nu e_q)}{1 - \nu^2},$$

$$q = \frac{E\,(e_q + \nu e_p)}{1 - \nu^2},$$

where

E = Young's modulus

ν = Poisson's ratio

The directions of the principal strains are given by:

$$\tan 2\,\theta_{p,q} = \frac{2e_2 - e_3 - e_1}{e_1 - e_3},$$

where $\theta_{p,q}$ is the direction of e_p and e_q as measured from e_1 and where e_p must make an angle of less than 45° with the algebraically greater normal strain e_1 or e_3.

If a 60° Rosette strain gauge is used, slightly modified formulas are employed for making the computations.

Before field tests were made at Rifle, Colo., laboratory tests were conducted at College Park, Md., during September and October 1947 to determine whether the above method was practical.

Two blocks of different materials were used. One was an oil-shale block 7x7x10 in. and the other a marble block of the same dimensions. An SR-4 electric strain gauge was cemented at the center of one of the faces of each block. Each block was individually loaded in a Baldwin Southwark compression table to simulate a face in a mine under compression. A prism section containing the strain gauge was sawed out and the strain relief measured with a strain indicator. The moduli of elasticity were de-

Fig. 7—Strain specimen being removed from the roofstone after sawing.
(Photograph by U.S. Bureau of Mines)

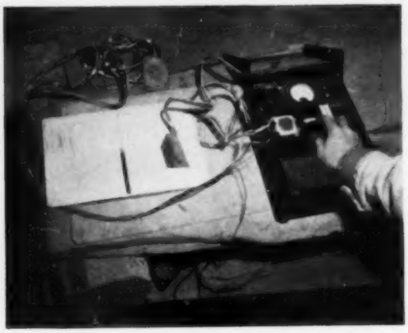

Fig. 8—Instrument for measuring expansion or contraction of strain specimen.
(Photograph by U.S. Bureau of Mines)

termined by standard methods and the stresses were computed and compared to the applied stresses. Remarkable agreement was noted. The following table shows the results:

Rock Type	Applied Stresses, Lb per Sq In.	Computed Stresses, Lb per Sq In.	Error, Pct
Marble	2430	2470	1.6
Shale	2040	1980	3.0

During 1948 and 1949 a number of field tests were made at Rifle, Colo., using practically the same procedure as that employed at College Park, Md., except that stressed mine faces were used instead of the stressed blocks. Locations for the tests were selected where the stresses at the test position could be computed from the overburden and shape of the opening. Measured stresses could then be compared to actual stresses. Several tests had to be discarded because of breakage of the specimen while being sawed out. Several others have not yet been computed because of the difficulty in obtaining the elastic constants. The results obtained in a 10-ft drift of nearly circular shape are shown in table II.

The results obtained in a long raise of square shape with rounded corners are shown in table III.

Table II. Results Obtained in 10-ft Drift of Nearly Circular Shape

Test Position	TP-1	TP-2	TP-3
Measured vertical compressive stress	680 psi	894 psi	1269 psi
Static vertical compressive stress	328 psi	328 psi	328 psi
Stress concentration factor due to shape of opening	2.1	2.7	3.8
Theoretical stress concentration factor due to shape of opening	3.0	3.0	3.0

Table III. Results Obtained in a Long Raise of Square Shape with Rounded Corners

Test Position	Measured Vertical Stresses, psi	Static Load, psi
TP-5	565	491
TP-6	71	491
TP-8	665	491
Avg	433	491
TP-10	59	538
TP-11	1148	538
TP-12	438	538
TP-13	734	538
Avg	595	538

Conclusion

As a direct result of the information obtained from the roof studies and the mine-structure stress analysis, openings in the Bureau's Experimental mine have been increased from 50 to 60 ft wide, leaving 60-ft-sq pillars to support the overburden.

It is planned to continue this phase of the research program and to improve the equipment methods and techniques so that more accurate data may be obtained. The test room is to be lengthened to 200 ft and widened in increments of 10 ft to determine the actual maximum safe roof span. Additional recording stations will be installed as the size of the room is increased.

Instruments developed for measuring mine-structure stresses will be installed in the Experimental mine to determine the roofstone behavior and the increase of pressure on the supporting pillars as the mine openings are advanced. From the information obtained over a period of time, it should be possible to predict what will happen in a large-scale commercial operation.

The equipment and procedures for roofstone studies and mine-structure stress analysis developed by the Bureau of Mines should find wide application throughout the mining industry in predicting safe mine-structure design.

Acknowledgments

E. D. Gardner, former Chief of the Oil-Shale Mining Branch, and Tell Ertl, former Chief of the Oil-Shale Mining Section, were instrumental in starting the research program. F. D. Wright, Mining Engineer, Bureau of Mines, and P. B. Bucky, Professor of Mining, School of Mines, Columbia University, and consultant to the Oil-Shale Mining Branch, completed most of the early laboratory studies. Full credit must be given to Leonard Obert, Physicist, College Park Station, Bureau of Mines, for equipment and technical advice. All members of the Oil-Shale Mining Section have contributed to the theory and application of the program.

An Unusual Test Of the Accuracy Of Well-Surveying Methods

by S. H. Williston

IT IS not often that bore hole surveys can be checked by actual civil engineering methods. A recent Arizona survey was checked by normal surveying methods and the comparison of the results should be of value to both oil and mining men.

During the summer of 1948 the Phelps-Dodge Corporation, at its Copper Queen property near Bisbee, Ariz., drilled a 1245 ft, 8 in. diam, churn drill-hole in a mineralized area and cased part of it, intending to use it to transfer mill tailings for stope fill. The hole, as frequently occurs, was not straight, and, in endeavoring to locate the bottom in the underground workings, they found no evidence of the hole at the underground coordinates directly below the surface location.

The noise of the drilling tools was reasonably clear, but the direction of sound was uncertain. Preliminary tests with available equipment were not successful in locating the bottom of the hole. Because of the mineralized character of the area and the fact there was some casing in the hole, any magnetic method of well surveying would give results of doubtful value.

Sperry-Sun gyroscopic well-surveying instruments were finally used to locate the bottom of the hole. These instruments consist of a gyroscope to determine azimuth and either a pendulum or bubble inclinometer. A multiple shot camera photographs both instruments on a single film and superimposes the photograph of a watch. Coordination of depth with time at the surface makes it possible to select the corresponding picture for any depth.

After making several runs of the empty instrument housing from the top of the hole to the bottom to make sure there were no obstructions in the hole, three surveys on wire line were completed during the afternoon. The three surveys, in which readings were taken at different points in the hole on each survey, were computed and gave the following locations of the bottom of the hole in relation to the surface collar: survey No. 1—24.92 N, 30.30 W; survey No. 2—24.24 N, 31.11 W; survey No. 3—26.54 N, 27.72 W. Then the data from the three surveys were combined into a single set of calculations which gave a location for the bottom of the hole: combined surveys—24.27 N, 30.16 W. (Fig. 1.)

Immediately upon the determination of the coordinates at the bottom of the hole, a drift on the 1300 ft level was started toward the indicated loca-

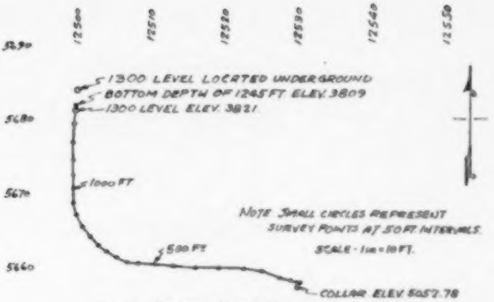

Fig. 1—Horizontal projection of survey churn drill hole No. 201.

Coordinates and Elevations

Location	Latitude	Departure	Elevation	
Collar	5657.71	12519.88	5052.78	Survey
Bottom (surveyed from top)	5681.98	12489.72		No. 1395
Depth of 1233 ft (surveyed from top)	5681.25	12489.64		
Depth of 1233 ft (located underground)	5684.16	12490.14	3820.86	

tion some 38 ft northwest of the coordinates of the surface location. The bottom of the hole was located within the drift round in which it was expected, and the transit survey run to the actual location of the hole indicated N 27.18, W 29.71. This shows a discrepancy between the well survey and the transit survey of 0.45 ft in the westerly direction and 2.91 ft in the northerly direction. All surveys, both gyroscopic and transit, fell well within the width of an ordinary drift. While this is satisfactory for almost any and all mining requirements, a theoretical examination was made as to reasons for the discrepancy.

A study of the course of the hole indicates that considerable right turn or spiral existed, and in all probability the surveying instrument was pulled out of alignment while traversing the turn by approximately 0.05 ft at the top and another 0.05 ft in the opposite direction at the bottom of the instrument. If such an allowance were to be made, the survey calculations would almost exactly correspond with those determined by transit. This sort of discrepancy would be minimized by the use of stabilizing guides.

It is unfortunate that physical laws probably effectively prevent the use of gyroscopic instruments in EX and AX diamond drill holes. The directive power of a gyroscope falls off inversely at some rate between the third power and the sixth power of the diameter. Present instruments can be run in casing 5¾ in. ID or over and might be adapted to somewhat smaller diameters, but the difficulty of reducing these diameters to 1¼ in. or 2 in. is almost insurmountable at the present time.

Acknowledgment

The author wishes to express his appreciation to the Phelps-Dodge Corporation for permission to publish this article, and to the Operating and Engineering Departments for their cooperation on the survey; also to Donald Hering, of the Sperry-Sun Well Surveying Co., who actually made the survey and calculated the results.

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AIME St. Louis Meeting, February 1951.

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Preliminary Report of Massco Circuitron

by Allen E. Craig, William J. Tait, and E. P. McCurdy

The Circuitron herein described applies current from the classifier motor circuit and energy from the sound of grinding media to move an oscillating disc. The disc through a photoelectric cell controls the speed of the ore feeder and consequently the amount of ore delivered to the mill.

THE Telluride Mines, Inc. at Telluride, Colo., is operating a flotation concentrator recovering lead, zinc, and copper with some gold and silver. The nominal capacity is 500 tons per 24 hr. The ore, which comes from several different mines, is characteristic of the district, changing from extremely tough rhodonite to soft gouge. The ore is crushed and washed, recrushed, and delivered to the mill bin at about ⅝ in. to serve as ball mill feed.

Details of Grinding Circuit

The material from the fine-ore bin is carried over a belt feeder driven by a Reeves vari-speed device. The belt discharges to a No. 86 Marcy ball mill, which operates in closed circuit, with jigs and a 6-ft Dorr classifier.

A Massco density controller holds the density of the classifier overflow at a predetermined point. The overflow, which contains about 5 pct + 48 mesh, goes to the flotation section.

Manual Operation

Before the adoption of automatic control for the grinding circuit, this work was all done manually by the operator. The bin gate was set to deliver 40 to 50 lb of ore per lineal foot of belt, and the speed of the belt was adjusted by the operator to deliver at the rate of about 500 tons per 24 hr. As the ore changed in hardness, specific gravity, or in the amount of moisture, or, as the size of the feed changed due to bin segregation, crusher setting, or other causes, the operator made corresponding changes in the rate of feed by manipulation of the Reeves control. Sometimes these changes occurred so rapidly that they caused sharp, pronounced surges, which created very considerable unbalance from normal conditions in the grinding circuit and, more critically, in the flotation section. Since it is not possible for the operator to anticipate or follow such changes closely, a very considerable unbalance from normal conditions may occur, which will require a drastic change in the rate of feed. These surges upset the fineness of grind and interfere radically with the balance between tonnage and reagent control. All of this tends to reduce the grade of concentrates, increase the loss in the tailing, and thus decrease the recovery of values.

The Circuitron

Some years ago, The Mine and Smelter Supply Co., in conjunction with the operators at the Telluride Mill, undertook to build a device that would give close control of the rate of feed from the ore bin and to coordinate with this a related device that would regulate the amount of sand load on the classifier rakes. Emphasis was strongly laid on the fact that such a control must apply to the entire grinding circuit and must include the feed to the ball mill, the classifier sand return, and the density of the classifier overflow. If this could be done successfully, we felt that the circuit could be operated under best possible conditions, with compensations

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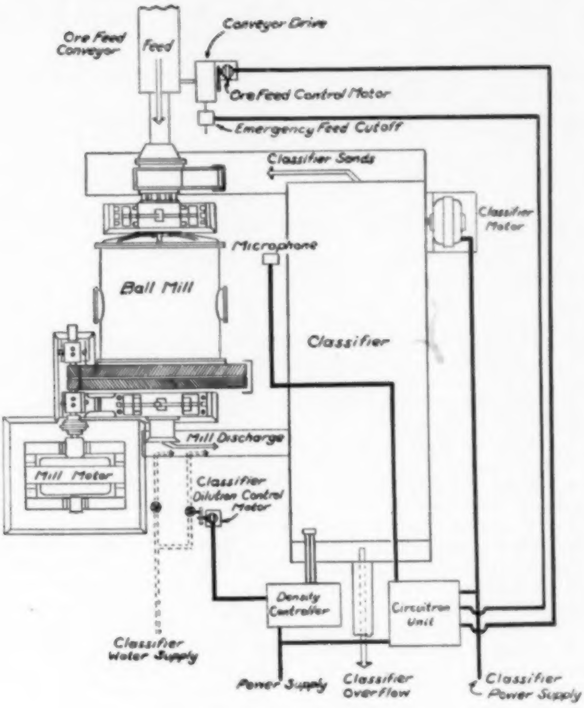

Fig. 1—Massco grinding circuit control.

being made continuously for changes in feed characteristics and feed rates. Further, we felt that if changes occurred, they could be noted and regulated before they became too great and, consequently, corrections could be made more evenly and gradually than could be done by manual control. Our desires in this respect have materialized, and the Circuitron has, in a very large manner, fulfilled our greatest hopes.

The Circuitron maintains constant control at all times so that as changes become necessary they are immediately noted and regulated before any great unbalance occurs. The control is so positive and the changes in regulation are made so gradually that surges are entirely eliminated. There are, indeed, occasions when the ore changes so quickly that the regular adjustments are too small for compensation, and there is a tendency to overload the circuit. Under such emergency conditions, our Circuitron actually stops the feed belt for a short interval, which may be from 5 to 80 sec. This complete cutoff allows the Circuitron to regain control but does not last long enough to interfere with the otherwise smooth operation. The tremendous advantage of this device is its positive control and its ability to make necessary adjustments so gradually that they are hardly noticed in the general mill operation. This is not possible with manual operation as it is now generally practiced.

We call this device the Massco Circuitron. Its position in the grinding circuit is shown in fig. 1. We have experimented for a long time and now have developed a control using electronic tubes, which gives very sensitive regulation and trouble-free service for long periods of time. Tube replacements can be made as required with a minimum amount of delay or effort. Such replacements become necessary only after months of service.

Control of Classifier

It is possible to imagine a condition in the grinding circuit in which the ball mill may be noisy and calling for more feed, but the classifier is loaded so that additional feed will be detrimental. There is no point in feeding more ore to the circuit because by so doing we simply increase the circulating sands or deliver a coarser product to the flotation section. So it is of vital importance to control the amount of sands returned from the classifier as well as to regulate the conditions existing within the ball mill itself. Naturally, the control of conditions, both in the ball mill and in the classifier, is based ultimately on the control of the amount of new ore delivered from the bin. When we control the classifier, we do, in fact, also affect conditions in the ball mill and, conversely, when we control the ball mill, we naturally affect conditions in the classifier. Nevertheless, two entirely separate sources of control are incorporated in our device to give control of the complete circuit.

We take the energy from the classifier motor through a transformer to the current coil of a wattmeter. The current delivered to this wattmeter will vary depending on the sand load on the classifier rakes. The wattmeter is modified to use a spring

tension and to reflect this variation in oscillation rather than in rotation of its disc. If the sand load on the classifier is constant, the wattmeter disc, under spring control, will oscillate as the rakes move over a definite arc an equal distance on each side of a central point. As the sand load increases or decreases, the swing will be greater to one side or the other side of the center point as the pull of the current in relation to the pull of the spring becomes greater or less.

Control of Ball Mill Load

If a microphone is placed near the shell of the ball mill, the sound of the balls striking on the steel liners will generate a small amount of audio energy. There is, however, interference from conflicting sounds such as that given by the ball mill gear, repair work in the vicinity, and from other outside sources so that, for our purpose, it is essential to narrow the range of the microphone to select only the desired sound of the balls on the steel liners. This is done by insulating and shielding the microphone with the result that a very small voltage, about 1/10 of a volt, is picked up and transmitted to an amplifier to give a resultant 100 v. The 1/10 v picked up by the microphone represents a sound too small to be distinguished by the human ear. By amplifying this to 100 v, we increase the sensitivity of the instrument so that the necessary adjustments and control can be made to a very fine point.

After amplification, the current is rectified and the direct current component is used to control the output of a generator tube, which delivers an alternate current voltage to the potential coil of the wattmeter. This voltage is proportional to the sound of the ball charge in the mill.

Operation of the Circuitron

The wattmeter now receives current from the classifier motor and voltage from the mill sound as transmitted through the microphone.

The wattmeter disc carries a steel shutter placed so that it will intercept a light beam which plays on a photoelectric cell. The photoelectric cell operates a two-pole, double-throw, heavy-duty relay. One section of the relay is used to change the direction of rotation of a reversible control motor, which operates the Reeves vari-speed drive on the feed belt delivering ore from the bin to the ball mill.

The circuit is adjusted so that when the light is acting on the photoelectric tube the reversible control motor will turn clockwise and increase the feed to the mill. When the shutter interrupts the light beam, the motor will turn counter-clockwise to reduce the feed rate. When the mill is operating properly, and no change in feed is required, the shutter will swing equal distances from the center so that the relay contacts are closed on one side and then on the other side and the Reeves control motor oscillates in balance.

When the ore becomes softer or finer, the mill becomes noisy, and the amount of voltage generated is increased. The torque, resulting from the increased voltage, is also increased to give a stronger pull against the wattmeter disc, and unobstructed light plays on the photoelectric cell for a greater part of the time to cause an increase in feed rate. As the mill becomes loaded again, and the voltage is reduced, the torque is reduced and the shutter swings back to intercept the light and reduce the feed.

In a similar manner, as the sand load on the classifier rakes increases, the amount of current drawn by the motor increases to pull the wattmeter shutter into the light beam and to reduce the feed. Conversely, as the classifier sand load lessens, the current is reduced and the spring tension pulls the shutter out of the light beam to increase the feed.

Overload Protection

The double-throw relay actuated by the photoelectric cell has another pair of contacts which are wired to a thermal tube. When the light beam is interrupted, this tube acts as an overload unit. If the mill becomes overloaded to such an extent that it cannot be controlled in the usual manner, with enough rapidity for safety, then this overload thermal unit opens the electrical circuit to stop the feed conveyor and actually cuts off the flow of ore to the mill for a few seconds.

Circuitron Operation

A typical log of the movement of the automatic control is tabulated below:

8:30 a.m.	Emergency cut-out 10 sec. Feed on 10 sec., off 30 sec., on again, in balance, for 5 min.
8:35 a.m.	Off 10 sec., on, in balance, 10 min.
8:45 a.m.	Coarse ore on the feed belt. Circuitron reducing feed.
8:50 a.m.	Feed still being reduced, off 55 sec.
8:53 a.m.	Still reducing.
8:55 a.m.	Off 5 sec., on 10 sec., off 20 sec., on and balanced.
9:05 a.m.	Fine ore on the belt-Circuitron increased feed.
9:10 a.m	Feed increased to same proportion as at 8:45 a.m.
9:15 a.m.	Control unit in balance.
9:18 a.m.	Reducing feed.
9:20 a.m.	Reducing feed.
9:21 a.m.	Feed off 15 sec., then balanced.

Conclusion

The length of time over which this device has been performing satisfactorily is comparatively short, and we have not had a full opportunity to gather any amount of operating detail.

We do know definitely that the Circuitron gives the following advantages: (1) Surges are eliminated, and necessary changes in feed rates are brought about gradually, (2) operating labor is reduced to a very marked degree, (3) tonnage is definitely increased.

From future study, we hope to gather data which will indicate the amount of tonnage increase. We hope, also, to prove an increase in the value of concentrate and a reduction in tail loss, thereby proving that the Circuitron will pay for itself in a reasonable time.

Acknowledgments

In the development of this Circuitron, we have had a great deal of valuable help from John Goetzcke, of the Mine and Smelter staff, and Olin J. Hurley in correlating the electronic principles to the mill operations.

We are deeply indebted to H. S. Worcester who, by his quiet insistence, kept us on this job and forced it through to the present state of success. We are also grateful for the whole-hearted cooperation and the patience exhibited by John Ferguson, Jr. and the entire operating force of the Telluride Mill.

Conductance Electrostatic Separation with Convective Charging

by Foster Fraas and Oliver C. Ralston

VIRTUALLY all commercial use of electrostatic separation has employed separators depending on differences of conductance of the broken, solid mixtures treated by them. The two main types of conductance separators have been: (1) Those in which the separating field is as near a static field between two charged electrodes as it is possible to maintain in the face of the usual leakage that takes place between electrodes with some 5 to 50 kv tension between them and spacing of ½ to 3 in.; (2) those in which a convective discharge of corona or spray type, but not a spark discharge, takes place between the carrier electrode and a charged electrode facing it, which is of very restricted area, such as a row of needle points or a small diameter wire parallel to the face of the carrier electrode. The present paper deals with this latter type. The principle is usable in separators where the carrier electrode, usually grounded, has the shape of a roller, a conveyor belt, a vibrating table, a sloping chute, or a nearly vertical plate. The innovation studied by us has been the use of the convective field for charging material on the carrier electrode and succeeded by a field between the carrier electrode and an electrode made of a true dielectric which cannot take on a charge by conduction but is charged by being in the radius of the corona discharge. The acquired charge is of the same sign as that of the corona but is not dischargable by contact with a good conductor. However, high-tension charge tends to leak off because of the ionization of the ambient air. The field between this electrode and the carrier electrode is as near static as it is possible to obtain. There are some differences in behavior of this electrode and that of the gas tube electrode used by Sutton, as will be discussed below. Both electrodes cause increased adhesion of nonconductive particles to the carrier roll. The particles are pinned there because they are convectively charged on their outer surfaces facing the corona electrode. The electrodes also cause increased rate of release of conductive particles tending to be pinned by corona charging but conveying the charge to the grounded carrier roll almost as rapidly as it is received by the particles.

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The sign of the potential on the corona-emitting electrode makes little difference in the separation. An electrode usually emits negative corona somewhat more easily than it emits positive corona, as electrons are then the fundamental particles emitted. With a positive electrode the flow of electrons involved in positive corona discharge is onto the electrode from the ambient gas.

Theory

In the separator illustrated in fig. 1, the nonionizing electrode is a rotating, nonconductive dielectric cylinder designated as *B*. The ions emitted by the corona electrode, or wire *F*, charge both the particles on the carrier electrode, or roll *A*, and the surface of the dielectric electrode *B*. Whereas the space between roll *A* and wire *F* contains ions, the space between electrode *B* and roll *A* is a pure static field free of ions.

The dielectric electrode has been described by Bullock.[1] However, the corona electrode, which Bullock uses for the purpose of charging only the dielectric electrode, is adapted here to also charge the particles on the carrier roll.

In operation the ionic field from the corona electrode is balanced with the static field from the dielectric electrode. The ionic field charges the particle so that it adheres to the roll. The static field intensifies the discharge of the corona charge by interfacial conduction and permits the release of the particle. The ionic and static fields therefore have opposing effects.[2] The values of both fields are adjusted so that the poor conductor will adhere and the good conductor will be released.

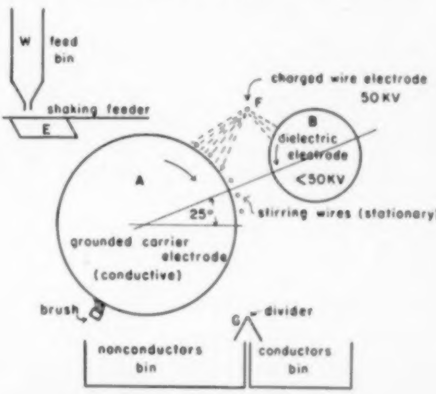

Fig. 1—Diagram of separator.

The intensity of both fields may be increased with this balanced adjustment, but the upper limit is frequently determined by the static field. With the dielectric electrode there is no spark discharge to interrupt the separation and no back deflection of conductive particles which may strike it. The dielectric electrode permits the establishment of a static field intensity that is higher than that possible with metal or conductive electrodes.

By increasing both the static and ionic fields under these balanced conditions, the efficiency of the electrostatic separator is increased. A proof of this follows.

The electrical action occurs in two stages, a preliminary stage in which the particles are charged by the corona, and a second stage where the particles in contact with the roll pass through the static field between the dielectric electrode and the roll (fig. 1). In the second stage the particles loose their corona charge at the rate:

$$\frac{\partial Q}{\partial t} = -(Q_r + Q_s)\,\frac{SC}{K}\,e^{\frac{-SCt}{K}}.$$

where:

Q_r = a constant charge proportional to F.

F = field intensity on the carrier electrode in the zone of separation resulting from the dielectric electrode.

Q_s = initial corona charge proportional to $r^2\sigma$.

r, K = particle radius and electrostatic capacity.

σ = surface density of corona charge.

S, C = interfacial area and conductivity.

t = time

$$Q = -Q_r + (Q_r + Q_s)\,e^{\frac{-SCt}{K}}, \quad \text{(Ref. 2)}.$$

The signs were determined so that at $t = o$, $Q = Q_s$, and at $t = \infty$, $Q = -Q_r$. During corona-charge loss, Q decreases from the value Q_s, passes through zero, and attains the value $-Q_r$. Q_r and Q_s have the same polarity. The final value of Q is of opposite polarity. The rate of loss of charge Q increases with the increase in Q_r and Q_s. Similarly the rate of change of Q with respect to the change in C increases with the increase in Q_r and Q_s.

$$\frac{\partial Q}{\partial C} = -(Q_r + Q_s)\,\frac{St}{K}\,e^{\frac{-SCt}{K}}$$

This relationship proves the statement that the *highest efficiency is secured when the ionic and static field strengths are balanced at their highest intensity.* In calculations the system of units must be considered.

The motion of a particle on the grounded roll is controlled by the opposing electrostatic and centrifugal forces f_e and f_c.

$$f_e = \frac{Q^2}{(2ir)^2} = br^2\sigma^2; \text{ outside of field } F$$

$$f_e = QF + \frac{Q^2}{(2ir)^2} = ar^2\sigma F + br^2\sigma^2; \text{ in field } F.$$

$$f_c = cr^3\rho\,(\omega^2 R + g\cos\phi)$$

where:

i = charge to image distance correction factor

ω, R = roll angular velocity and radius

ρ = density of particle

a, b, c = proportionality constants which include shape factors

ϕ = angle between verticle and line through particle and shaft center line

g = acceleration of gravity.

The equation for fc would be modified with other types of carrier electrodes.

In the field F, the corona-charged particle has a greater adhesion to the roll than out of the field. When the particle is in the field but off the roll, the attraction toward the roll is less than when the particle is on the roll. This is an explanation for the violent ejection of the conductors from the roll. A great force of adhesion, if present momentarily with the conductive particles, is a factor contributing a favorable interfacial surface conductance.

If for a given particle size the separator is adjusted so that for the nonconductive particle $f_e = f_c$, then for all larger particles, $f_e < f_c$, and for all smaller particles, $f_e > f_c$. The correct procedure is to select an adjustment so that $f_e \geqq f_c$ for the maximum nonconductor particle size. Actually this adjustment is made relative to the position of the dividing edge (G, fig. 1) and not with respect to release from the roll.

Table I. Beneficiation of Hematite Ore

Fraction	Weight, Pct	Assay by Particle Number, Hematite, Pct	Chemical Assay, Fe, Pct	Total Fe, Pct
Concentrate	82.7	97	67.7	90
Rejects	17.3	19	35.9	10
Composite	100	78	62	100

Table II. Size Analysis of the First Roll Hematite Concentrate

Fraction Mesh Size	—20 + 35	—35 + 65	—65 + 100	—100	Total
Weight, Pct	42.7	35.7	11.2	10.4	100

In the separation of a mixture of a conductor and a nonconductor where the conductivities are highly divergent, all the particles of the conductor, both coarse and fine, will lose their corona charge and be deflected before the nonconductor particles are released. This is illustrated with crushed quartz-pyrite.

When the conductivities are not highly divergent, the corona charge loss rates from the two constituents become competitive. Such a condition arises even though there is still a large difference in the specific conductivities of the compact crystals of the two constituents. This results from the probability factor in the conductance at the particle-roll interface. Measurement has shown that the electrical charges acquired by galena particles on a copper carrier roll with a static field are distributed as a logarithmic function of the normal probability law. Galena is a comparatively good conductor. The charges were measured as each particle fell through a Faraday cage, which was connected to an FP 54 thermionic electrometer tube, a direct current amplifier, and a high-speed recorder.

A frequency distribution of charging rates may result from either variations on each particle or variations between particles. The former has been demonstrated with straight-line probability plots. With the latter, some particles may have, by chance, a low conductance surface exposed. Regrinding would yield a new distribution.

In general, all factors that may improve the interfacial surface conductance, such as good contact and the cleaning of the carrier roll, have an influence on the efficiency of the separation.

When the conductivities are not highly divergent and the adjustments are made so that, $f_c \leqq f_n$, *a high purity cut can be taken of the good-conductor fraction but not of the poor-conductor fraction.* Fine good-conductor particles will always be found with the coarse and fine poor-conductor particles. *Size classification is required for removing a high purity cut of the poor conductor.*

Application

Although the performance of the separator may be illustrated with a variety of ores, the results with those containing locked particles is most significant.

The repulsion effect of the high intensity static field on the corona charged nonconductors is sufficient to cause the firm adhesion of locked particles of conductor and nonconductor to the carrier roll. The corona-charged nonconductor portion cannot be discharged, although the conductor portion may be in contact with the roll. Such a response has previously been suggested by J. L. Gilson.

The poor-conductor fraction will contain therefore all the locked particles. Since the separator capacity is highest at the coarsest size, it is good practice to separate initially as much of the coarse good-conductor as possible. A most efficient separation is secured by the following procedure:

1. Remove unlocked good-conductor particles at the coarsest size.

2. Grind the undeflected residual mixture of locked particles, nonconductor particles and low-probability good-conductor particles to finer sizes, and repeat the separation.

Table I and fig. 2 summarize the results with a quartz-hematite (specular) ore from Lyon Mountain, N. Y. The carrier roll and dielectric electrode had diameters of 6 and 3 in., respectively, and rotation speeds of 150 and 300 rpm. A simple form of the dielectric electrode consists of a methyl methacrylate plastic tube (0.13-in. wall thickness) placed as a close fit over a precision metal tube. Corona wires having diameters from 0.003 to 0.01 in. and potentials up to 100 kv are satisfactory.

The feed rate was 0.15 ton per hr per ft length of roll. Higher rotation speeds and feed rates up to 0.25 ton per hr per ft length of roll were used with some sacrifice in grade and recovery. The feed and the separator were heated to counteract the effects of high humidity.

Laboratory batch tests were conducted on a single roll separator with the subsequent products carried through the flowsheet steps of fig. 2. Here a single roll will remove a large portion of the unlocked particles of hematite from the —20-mesh grind. Another single roll removes the unlocked particles from the —35-mesh and another two rolls most of the particles from the —65-mesh grind.

The corona electrode at the first unit was close to the carrier roll. On the succeeding units, it was

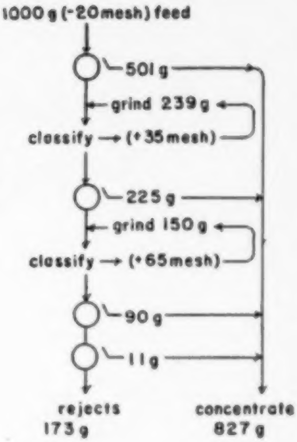

Fig. 2—Flowsheet for hematite ore.

withdrawn from the roll as the particle size decreased. The dividing edge setting at the last three units was close to the roll, and at the first unit, far from the roll. Although a small proportion of low-probability conductors depends on particle size, the major portion of the feed is removed with coarse and fine sizes combined. This is shown in table II by the size analysis of the product from the first roll.

An important feature of this separator is that the product from each roll is a finished concentrate requiring no recleaning.

Ilmenite, high-iron chromite, and magnetite have conductivities which correspond to specular hematite and yield equivalent results. Low-iron chromites have lower conductivities, and when associated with diopside and serpentine having apparent conductivities higher than that of quartz, size classification of the fine fraction is required. Commercial operation would require air or an electrostatic method for classification at sizes finer than 20-mesh. Electrostatic sizing will be discussed in a later paper.

The 0.018-in. diam steel stirring wires shown in fig. 1 were not used in the test described in this paper. They increase the separation efficiency with rutile-zircon mixtures and are more effective with fine than with coarse material. The principle involved is repeated and higher pressure contact of the particles with the roll. Vibration of the carrier electrode or periodic interruption of potential applied should have a similar effect.

A separator is described in which a dielectric electrode and the particles on a carrier roll are simultaneously charged by corona. The characteristics and the procedure for operating the separator on step crushed ores are detailed.

Acknowledgment

Acknowledgment is given to P. M. Ambrose, chief of the College Park Station, and to F. D. Lamb, in charge of the mineral-dressing unit, for administrative guidance, and to H. W. Jaffe and A. H. Macmillan for petrographic and chemical analysis studies.

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Pilot-Plant Investigation of Concentration of Blackbird Cobalt Ore by Roast-Flotation Process

by S. R. Zimmerley and S. F. Ravitz

High-grade cobalt concentrates were produced from the complex Blackbird ore with very good recovery in continuous pilot-plant operations in which a low-grade bulk cobaltite-pyrite flotation concentrate was roasted in a multiple-hearth furnace to oxidize the pyrite selectively. The cobaltite was floated from the resulting calcine.

THE existence of the cobalt-copper deposits in the Blackbird district, Lemhi County, Idaho, has been known for many years. However, these deposits have not yet been exploited successfully because of the low grade of the ore, absence of information on ore reserves, and lack of feasible methods for concentrating and recovering cobalt from the ore.

In 1941, when it appeared that World War II might interfere seriously with imports of cobalt from overseas, the U. S. Bureau of Mines started an extensive mining and metallurgical investigation of the Blackbird deposits. Exploration by trenching and drilling[1] indicated that ore reserves were adequate to warrant consideration of large-scale exploitation. A process for producing high-grade cobalt concentrates from the ore was developed at the Salt Lake City Station of the Bureau,[2] and hydrometallurgical procedures for producing cobalt metal or oxide from the concentrates were developed at the Boulder City Station.[3] The Howe Sound Co., through a subsidiary, the Calera Mining Co., carried out further exploratory work, which increased probable ore reserves to a tonnage commensurate with a large investment in mine, mill, transportation facilities, and refinery. Simultaneously, the metallurgical staff of the company, working in cooperation with the Bureau of Mines, further investigated the metallurgical problems. It was the opinion of both groups that pilot-plant testing was required to translate laboratory results into plant operation.

In general, the Blackbird ore has a high content of pyrite and some pyrrhotite and contains 0.5 to 1 pct cobalt and 1 to 2 pct copper, which are present essentially as cobaltite and chalcopyrite. The copper can be recovered readily as a high-grade concentrate by selective flotation. The cobalt then can be recovered as a bulk sulphide concentrate assaying 4 to 5 pct cobalt, but no satisfactory procedure for separating the cobalt from the pyrite and pyrrhotite directly by flotation had been found.

The method developed by the Bureau for producing a high-grade cobalt concentrate is essentially an application of the old Horwood process of selective roasting.[4] The bulk sulphide concentrate is roasted to form an oxide coating on the iron mineral particles, and the calcine is then treated by flotation. The oxide coating prevents the iron minerals from floating so that it is possible to obtain a high recovery of cobalt in concentrates assaying more than 20 pct cobalt.

Laboratory tests by metallurgists of both the Bureau and the Howe Sound Co. had indicated the optimum roasting temperature to be 425° to 450°C. At lower temperatures the rate of oxidation of the iron minerals is too low, whereas at higher temperatures substantial quantities of cobalt are rendered water-soluble. Because of its high sulphur content (35 to 40 pct), the bulk concentrate tends to continue to burn after it has reached the optimum roasting temperature. Although excessive temperatures could be avoided in the small-scale laboratory furnaces, there was considerable doubt that the temperature could be controlled properly in industrial equipment.

To provide material for the pilot-plant tests, the Calera Mining Co. installed a small, temporary flotation mill at the Blackbird property and treated

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Table I. Results of Continuous Campaign

Period	Feed Rate, Lb per Hr[a]	Time, Hr	Calcine Assay, Pct			Sulphur Eliminated, Pct[b]	Co Assay, Pct			Co Distribution, Pct[c]		
			Co	Fe	S		Conc.	Mid.	Tail.	Conc.	Mid.	Tail.
Roasting Bulk Concentrate: 4.5 pct Co, 36.6 pct Fe, 38.9 pct S												
1	118	24	5.1	40.7	23.9	45	23.0	5.3	0.65	80.9	10.9	8.2
2	170	28	5.1	40.1	27.9	35	21.7	2.9	0.40	87.3	7.0	5.7
3	187	52	5.0	40.3	30.3	29	17.7	3.2	0.40	87.0	8.1	4.9
4	195	92	5.0	39.8	29.4	30	17.8	3.2	0.38	86.7	8.5	4.8
5	169	64	5.0	41.5	26.7	39	20.7	4.9	0.50	82.1	11.3	6.6
6	172	76	5.0	41.9	25.7	42	21.7	4.3	0.45	84.8	9.8	5.4
7[d]	172	40	5.4	37.0	20.4	48	21.5	4.3	0.62	82.0	10.3	7.7
Roasting Underroasted Calcine: 5.5 pct Co, 40.0 pct Fe, 34.5 pct S												
8	167	82	5.2	43.0	22.5	51	22.9	4.5	0.43	87.6	6.3	6.1
Roasting Flotation Middlings: 4.2 pct Co, 40.9 pct Fe, 31.5 pct S												
9	169	61	4.6	44.0	23.2	50	21.9	4.9	0.58	81.3	8.8	9.9

[a] Including recycled flue dust, which constituted 6 to 12 pct of feed in all periods except No. 7.
[b] Calculated from iron and sulphur assays of bulk concentrate and calcines.
[c] Exclusive of dissolved cobalt in mill solutions (0.6 to 2 pct of total cobalt).
[d] Feed consisted of approximately 38 pct bulk concentrate and 62 pct flue dust.

350 tons of ore, producing about 50 tons of bulk cobalt concentrates assaying approximately 4.5 pct Co, 37 pct Fe, 39 pct S, and 0.45 pct Cu, which were shipped to the Salt Lake City Station of the Bureau.

Pilot-Plant Equipment and Procedure

Roasting: The bulk concentrate as received was too wet and lumpy to feed uniformly to the roasting furnace. Therefore, it was air-dried and screened through 10-mesh. The +10-mesh lumps were allowed to air-dry for a day or two and then were crushed and returned to the screen.

The bulk concentrate was roasted in an experimental Herreshoff furnace, which was 6½ ft high, 3 ft in diameter inside the refractory lining, and equipped with eight roasting hearths. The effective hearth area was 49 sq ft, and the time required for the charge to pass over each hearth was 10 to 15 min. The actual temperature of the material on each hearth was determined by means of a bare-junction thermocouple inserted by hand directly into the charge. The furnace gases passed first through a cyclone dust collector, then through a venturimeter for measuring the volume, and finally through a fan to the stack. The flue dust was mixed with incoming bulk concentrate and returned to the furnace.

Flotation: The calcine was fed at the rate of approximately 300 lb per hr to a conditioning tank, where it was mixed with water to give a pulp density of about 30 pct. An aqueous emulsion containing 0.4 to 0.5 lb Minerec A and 0.05 lb B-23 (stabilized with 0.03 lb Syntex M) per ton of calcine was added at the conditioner. The pulp then flowed through the mechanically agitated flotation cells. The first two cells produced a rougher concentrate, which was cleaned in a third cell, the cleaner tailing being returned to the first rougher cell. The rougher tailing was retreated in a scavenger cell to produce a middling product and a tailing; approximately 0.1 lb of additional collector and 0.05 lb B-23 per ton of calcine were used in the scavenger circuit. The middling product, which had approximately the same cobalt assay as the original bulk concentrate, was air-dried and stored for reroasting.

Flotation was carried out at the natural pH of the pulp which, with properly roasted calcine, was 3.5 to 4.0. The mill solution also contained 0.15 to 0.4 g of cobalt, up to 0.2 g of copper, and about 4 g of iron per liter. Very sensitive chemical tests were made for arsine and hydrogen sulphide in the exit air from the cells, but none was detected.

Experimental Results

To obtain operating information needed for a continuous campaign, the furnace was operated intermittently for the first two weeks. It was found that burning of the charge could not be confined to the upper hearths by ordinary furnace manipulation. Burning gradually advanced downward so that the calcine was discharged at 450°C or higher and continued to burn in the storage piles.

Laboratory tests on various portions of calcine showed that if the calcine is either quenched from roasting temperature or allowed to cool in the absence of air, virtually no selectivity is obtained in flotation. Above 425°C the rate of oxidation of the first atom of sulphur in the pyrite is apparently much greater than that of the second so that the surfaces of the particles consist essentially of FeS (or artificial pyrrhotite) and unaltered cobaltite, both of which float readily. However, if the burning calcine is cooled slowly to below about 320°C, burning ceases and the FeS is oxidized to iron oxide, as evidenced by a change in color from gray to red. The pyrite particles thus are made nonfloatable, and very good selectivity is obtained in flotation.

In view of these results, it was decided to control the cooling entirely within the furnace by means of water sprays in order to maintain a maximum charge temperature of about 450°C and a discharge temperature below 300°C. The furnace was rabbled out, and feeding was resumed at a relatively low rate. By supplying a small amount of heat at hearth 3 and injecting a fine spray of water at intervals as needed on hearths 4 to 7, the following average temperature distribution in the material on the various hearths was readily maintained: hearth 2, 408°C; 3, 459°; 4, 456°; 5, 419°; 6, 414°; 7, 340°; and 8, 265°. Introduction of air into the furnace was controlled by means of the doors

on the two bottom hearths; all the other doors were kept closed.

This method of operation immediately gave very good flotation results. The furnace was operated continuously in this manner for the duration of the campaign, about 1 month. The feed rate was varied at intervals to determine the optimum furnace capacity. Toward the end of the campaign, under-roasted calcine produced in the preliminary operations and the accumulated middling product from flotation were roasted. The flotation mill, which had a somewhat higher capacity than the furnace, was operated an average of about 12 hr per day. The data obtained are summarized in table I and are discussed below. The nine periods listed were consecutive and cover the entire continuous campaign.

Treatment of Bulk Concentrate

The grade of concentrate was 23 pct cobalt at the minimum feed rate of 118 lb per hr (period 1), dropped to 21.7 pct when the feed rate was increased to 170 lb per hr (periods 2, 5, and 6), and dropped further to slightly less than 18 pct at a feed rate of about 190 lb per hr (periods 3 and 4). Sulphur elimination from the bulk concentrate averaged 45, 39, and 30 pct, respectively, for the three feed rates.

In general, 81 to 88 pct of the cobalt was recovered in the concentrate at a grade of 18 to 23 pct; 6 to 11 pct was recovered as a middling product assaying 3 to 5 pct; and 5 to 8 pct was lost in the tailing, which assayed 0.4 to 0.65 pct. In addition to the tailings loss, 0.6 to 2 pct of the cobalt was lost as dissolved cobalt, as determined by occasional analyses of the mill solutions.

At the beginning of the continuous campaign, the only spray nozzle available was one that produced a conical spray so that some of the water impinged directly on the charge. At the start of period 6, this nozzle was replaced by one that formed a fan-shaped mist so that much less water came in contact with the charge before vaporizing. An increase of 1 pct in grade in period 6 as compared to period 5 may have been caused by this change. The average quantity of water used throughout the continuous campaign was 220 lb per ton of feed.

Flue Dust

Although most of the flue dust was recirculated to the furnace with incoming bulk concentrate, a considerable quantity was on hand at the end of period 6, when nearly all the bulk concentrate had been roasted. This dust was mixed with the remaining bulk concentrate, and the mixture, consisting of 62 pct flue dust and 38 pct bulk concentrate, constituted the feed for period 7. The data in table I show that this feed gave nearly the same results as had been obtained previously. However, 22 pct of the weight of the feed during this period was collected as flue dust in the cyclone.

Treatment of Flotation Middling

By the end of period 8, nearly 5 tons of middling flotation product had been produced. This product had an average analysis of 4.2 pct Co, 40.9 pct Fe, and 31.5 pct S, and thus was similar in analysis to the original bulk concentrate except for its somewhat lower sulphur content. It constituted the feed for the last period of operation of the pilot plant and gave a very good grade of concentrate—21.9 pct. Recovery in the concentrate, 81 pct, was slightly less than in the treatment of the bulk concentrate; loss in the tailing, 9.9 pct, was slightly higher.

Conclusions

The following data for period 6 are probably representative of the results that would be obtained in plant practice. Taking into consideration the soluble cobalt loss (0.6 pct), about 84 pct of the cobalt was recovered at a grade of nearly 22 pct, 10 pct was recovered as a middling product, and 6 pct was lost in the tailing and in the mill solutions. On retreatment of the middling, about 9 pct of its total cobalt content, or slightly less than 1 pct of the cobalt in the original bulk concentrate, was lost; the remainder eventually would be recovered at a grade of approximately 22 pct. It is probable, therefore, that an over-all flotation recovery of 93 pct of the cobalt at a grade of about 22 pct could be expected in commercial operation.

The total feed rate to the furnace during period 6 was 3.5 lb per sq ft of hearth area per hour. On this basis, the capacity of a 21-ft, 8-hearth furnace would be approximately 100 tons of feed per 24 hr. Assuming a 20-pct circulating load of flue dust and middlings, the net capacity would be about 80 tons of bulk concentrate per day.

The flotation cell capacity (including cleaning) was 1.9 tons of calcine per cu ft per 24 hr. Allowing for a 15-pct loss in weight during roasting and a 10-pct circulating load of middlings, the net flotation capacity in terms of original bulk concentrate would be about 2 tons per cu ft per 24 hr.

The results of this investigation indicate that the roast-flotation process is technologically feasible for treatment of the Blackbird ore and that the principle of preferential oxidation induced by heat may warrant consideration for the preparation of other complex and refractory ores for beneficiation by flotation.

Acknowledgments

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Kerosine Flotation of Bituminous Coal Fines

by L. E. Schiffman

This paper describes the operation of two kerosine flotation plants in Alabama for cleaning —10 mesh bituminous coal. One plant treats washer sludge, the other raw coal. Data on capacity efficiency and capital and operating costs is given. Methods for increasing cell capacity are discussed.

IN cleaning coal it has long been recognized that methods which give excellent results for the coarser sizes may give poor or even no cleaning for the finer sizes. Effective cleaning of the fines, therefore, is usually a separate and distinct problem. For many years the problem of fines was considered of relatively minor importance. Now there is growing interest in methods of cleaning the fine sizes of coal because:

1. There are more fines. Increasing mechanization and the adoption of full seam mining methods have resulted in the production of a greater percentage of smaller particles than were produced by hand loading methods.

2. The fines have deteriorated in quality. The same causes that have led to increased quantity have also operated to decrease quality through greater contamination with slate, rock, and other diluents.

3. The fines have become more valuable. When coal was comparatively cheap, loss of the finer sizes was tolerated but, with the rapid increase in the cost of mining coal, it has become imperative that the maximum amount of clean coal be recovered from the mixture of coal and rock delivered to the tipple.

4. In many cases, the fines have become a stream pollution problem. Increasing governmental activity and legislation to prevent stream pollution often have made it necessary to retain the fine material which formerly was permitted to escape with the washer effluent into flowing streams.

The Sloss-Sheffield Steel and Iron Co. operates four coal mines, all within a 20-mile radius of Birmingham, Ala., with a total annual production of 1,250,000 tons of washed coal. Three of these mines are in the Mary Lee seam and one in the Jefferson seam. Although some steam and domestic coals are produced, the major portion of the coal is used to make coke for blast furnace operation. Fine coal is desirable for coking, and all the coal so used by Sloss is crushed to ¼ in. Therefore, there is no problem of an outlet for even the finest sizes of coal. Since the coal is used for metallurgical purposes, low ash is of greater importance than would be the case if it were used for steam generation or domestic heating. Based on present blast furnace practice at Sloss, it is calculated that each percentage of ash in the coal carries a penalty of $0.22 per ton of coal. Thus there is a considerable margin for operating costs in a fine coal cleaning method that will result in materially lowering the ash content of the cleaned coal.

Development of Fine Coal Cleaning Problem

In 1942 Sloss reopened its Bessie mine and constructed at this location a new preparation plant to clean 1¼ in. x 0 coal from the Mary Lee seam.

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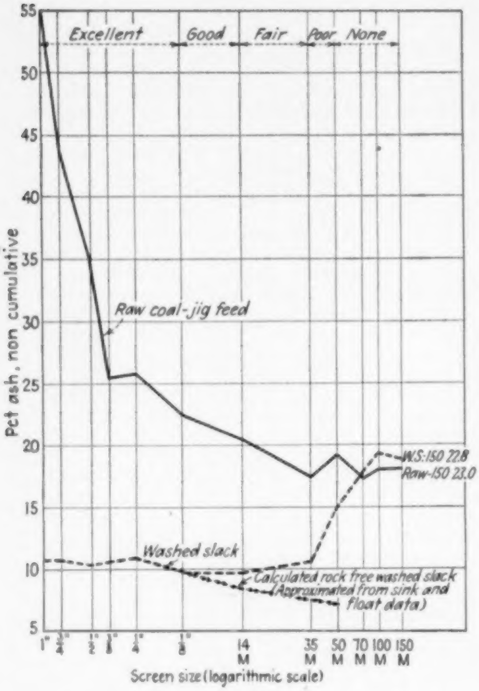

Fig. 1—Separation of coal from rock made by jig.

Ash content of sized coal. Screen analyses of Flat Top coal, average samples Dec. 1 to 8, 1944.

A McNally Norton (Baum type) five-compartment primary jig and a similar two-compartment secondary jig were installed. Contrary to the company's previous practice, the installation included 1-mm screens for dewatering the washed products and a sludge settling tank for recovering the solids in the screen underflow. With this plant in service, the —1-mm fines became separately available prior to mixing with the coarser coal. Samples analyzed for ash then revealed a fine coal cleaning problem because, with the coarser coal having an ash content of about 10 pct, the sludge ash was about 15 to 20 pct.

In 1943 Sloss installed new washing equipment at its Flat Top mine to clean 1⅛ in. x 0 Mary Lee coal. Jigs duplicating those just installed at Bessie replaced six American (plunger type) jigs. Unlike Bessie, dewatering equipment for the primary coal was not included. Contrary to expectations, when the new equipment went into service, the ash in the primary coal was higher than it had been with the old equipment. Six months of investigation, testing, and adjustment followed. At the end of this period the ash was not appreciably lower, but the cause for the higher ash had been determined. In the old plunger jigs the fine coal, rock, and slate found their way into the hutch and, since the hutch discharge was wasted, did not appear in the finished product. In the new primary Baum type jig, the fines, including that portion of the rock and slate not removed by the jig, remained in the washed coal. Sink-and-float tests together with screen analyses of the jig products indicated that down to 14-mesh the jig did a good job, down to 35-mesh, a fair job, and below 35-mesh, a poor job. The effectiveness of the jig in turning out a slate-free product is shown in fig. 1. The —35-mesh coal is about 10 pct of the total primary coal. Even with the increase in ash, it is obviously an advantage to retain this amount of coal that previously had been largely wasted, but, with the acquisition of these added coal fines, there also was acquired a fine coal cleaning problem.

Experimental Cell Installations, Bessie

Thus it became established definitely that below a size of about 1 mm the jigs were not making a good separation and that some better method was needed for cleaning coal below this size. Meanwhile, B. W. Gandrud, of the Bureau of Mines, had discussed with Sloss some of the very interesting results he had secured in laboratory experiments using kerosine flotation for fine coal, and the Sloss-Sheffield Co. became interested in proving the feasibility of the method on a larger-than-laboratory scale.

In 1943, collaborating with Gandrud, Sloss built and installed at Bessie mine a homemade flocculation cell for the flotation of the sludge coal. This cell is shown in fig. 4*a*. Bessie was chosen as the place for the experimental work because the fine coal was already separately available in the form of sludge from the settling tank. The homemade cell clearly demonstrated that a good separation could be made. The results were sufficiently encouraging to warrant further experimentation. In November 1943, a 2-cell No. 24 (43x43) modified Denver Sub A flotation unit was installed. The modification consisted of a raking device to remove the floated product in place of the conventional paddle for froth removal. The 2-cell unit gave a suitable floated product of about 8 pct ash content, but the refuse leaving the cells was quite low in ash. It became apparent that more than two cells in series would be required if loss of coal in the refuse were to be held to a reasonable amount. Accordingly, an additional 2-cell unit was installed. With the four cells in series, the flotation equipment was put into continuous service, handling about 10 to 15 pct of the sludge, and became a part of the regular washer operation. Under this arrangement, extensive testing was possible. The results

Table I. Bessie Operating Statistics, Average of Daily Results

Seam, Mary Lee; output, 1,600 tons coking coal per day and 125 tons secondary coal per day.

	With Flotation 12 Months Ending Aug. 31, 1949	Prior to Flotation 6 Months Ending Oct. 31, 1947
General Washer Results		
Raw coal 1¼x0, ash	31.9	30.4
Washed coking coal 1¼x0, ash	10.1 [a]	11.4
Secondary coal, ash	17.3	13.3 [b]
Float in jig refuse at 1.45 sp gr, pct	3.0	3.2
Flotation Details		
Raw sludge, ash	20.0	
Floated coal, ash	8.2	
Secondary coal, 0' size from reject, ash	24.6	
Refuse from flotation, ash	55.5	
Reagent, 5 lb kerosine per ton of feed		

[a] Includes floated coal.
[b] Secondary coal 275 tons per day compared with 125 tons in first column.

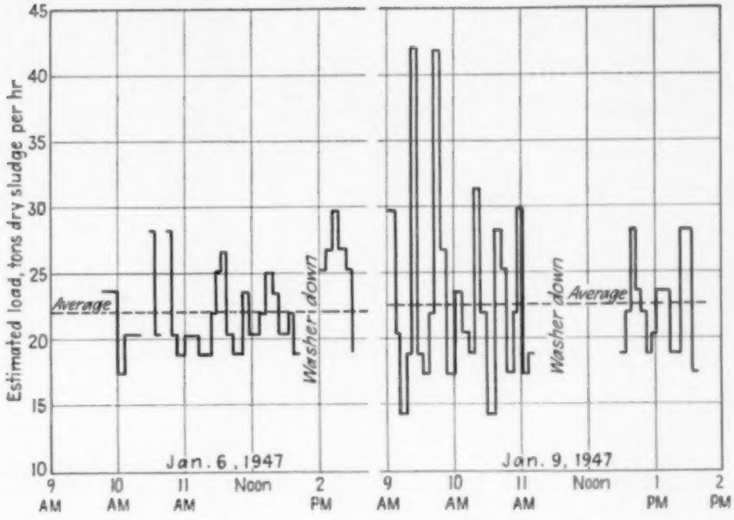

Fig. 2—Estimated loading of sludge recovery drag, Bessie mine.

Estimate based on observation of volume at 5 min intervals. One cu ft of sludge on drag taken as 54 lb dry sludge. Drag speed was 14 flights per min.

secured from the 4-cell operation were good, with a refuse running 59 pct or higher in ash content.

Flotation of Sludge, Bessie Plant

A year of operation on a small portion of the sludge proved the practicability of the process. It then was decided to install equipment of sufficient capacity to process all of the sludge. As a preliminary step, a study was made of the rate at which the sludge was being removed from the settling tank by the drag conveyor used for this purpose.

It was found that the average quantity of sludge was about 22 tph, but the rate of withdrawal fluctuated widely, reaching a peak rate of about 45 tph. The fluctuations in rate are shown in fig. 2. Therefore, it was decided to make the installation of such capacity that it would handle 45 tph with a sacrifice of some coal in the refuse and 22 tph with but little loss of coal. Accordingly, two more 2-cell No. 24 (43x43) Denver "Sub-A" units and two 2-cell No. 30 (56x56) Denver "Sub-A" units were installed, making a total installation of 12 cells. This equipment was put in service in October 1947 and has been in continuous service since that time.

The Bessie washer flowsheet is shown in fig. 3 so that no attempt will be made to describe it here. Suffice it to say that the —1 mm coal from both the primary and secondary circuits is delivered to the sludge settling tank from which it is removed by a slow-moving drag conveyor and then is discharged into a stationary splitter which divides it into two equal parts. The cells are installed in two parallel rows, each of which receives one half the sludge. Each row consists of two 100-cu ft cells (No. 30) followed by four 50-cu ft cells (No. 24) all in series. Floated coal is removed by a single overflow raking mechanism (replacing the usual froth paddle) in each cell and is discharged into two dewatering screws, each of which runs the length of a cell row with the discharge end of each screw projecting over the drag conveyor that carries the jig-plant washed coal to the loading bin. These screws serve the dual purpose of conveyors and dewatering devices. The drainage from the dewatering screws is returned to the first cell in each row by gravity through a side entrance in the cell. The unfloated material, after having traveled through the six cells in series, is discharged into a 6-ft dewatering cone to allow the coarser portion to settle.

The cone underflow is pumped to a shaking screen equipped with ½-mm screen cloth, and the +½-mm material is recovered and thrown into the secondary coal. The underflow from the screen joins the overflow from the cone, and these two products, constituting the refuse from the flotation plant, are sent to waste. There is no prior conditioning of pulp before entrance to the cells. The entering pulp averages about 24 pct solids by weight. Reagent flows by gravity from a 6000-gal storage tank through a cup feeder and a rotating distributor into the pulp streams at the splitter (75 pct) and into the inlets of the fourth cells (25 pct). The flotation plant receives an average of about 22 tph of sludge containing about 20 pct ash. In the plant this 22 tons is divided into approximately 14 tons of floated coal of 8 pct ash content, which goes into the primary coking coal, four tons of oversize reject of 24 pct ash content, which goes into the secondary coal, and 4 tons of refuse of 56 pct ash content, which goes to waste. Originally the reagent used was a mixture of kerosine and B23 frother, but at present only kerosine is used. Approximately 5 lb of reagent are used per ton of feed, which is greater than laboratory work by Gandrud has shown to be needed. However, considering the fluctuating character of the feed, the reagent requirements are not excessive. Table I gives a summary of operating results at Bessie mine over a period of 12 months ending August 31, 1949. Mine operation since then has been sporadic and later results are not included. Table II gives typical screen analyses of cell feed and resulting products.

Prior to the installation of the flotation plant, the Bessie washer was operated by two men, a jig operator and a helper. Because of the simplicity of the kerosine flotation process, the cell operation re-

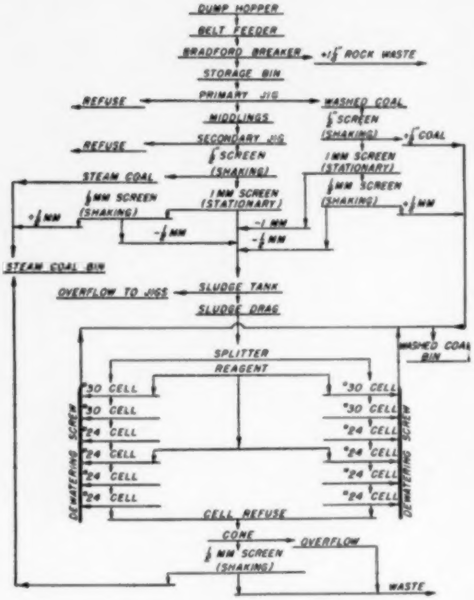

Fig. 3—Flowsheet, Bessie washer.

quires no additional labor. The helper attends to the flotation units on a part-time basis and devotes the rest of his time to oiling washer equipment and assisting the jig operator. Since there is no accurate means of measuring feed to the flotation units, cost data is necessarily approximate. The data given is based on occasional measurement of feed by weighing the sludge on several flights of the drag conveyor and by volumetric measurements on other flights and also on calculations arrived at from pulp densities and water flow. Also, because of the comparatively brief length of time the plant has been in service, maintenance costs are not yet fully known. The maintenance cost given is based on renewal of the wearing parts of the cells every two years plus maintenance expense already incurred, which has been largely the upkeep of the raking mechanism. Table III gives the estimated operating cost of the Bessie flotation plant.

When plans were made for the Bessie installation, it was thought that it would be necessary to eliminate fluctuations in the cell feed to secure satisfactory operation, and money for this purpose was included in the appropriation request for modification of the sludge settling tank. For the first year of operation, the cell feed was substantially as shown in fig. 2. The units proved to have a remarkable capacity for absorbing the peak feed rates which, as can be seen, are of comparatively short duration. Reagent was set for average rather than maximum rate, and the cells delivered a floated product at a rate much more uniform than the feed would indicate. About a year ago, a minor modification was made in the settling basin, which has to a degree reduced the swings in cell feed, but the feed is still far from uniform. It seems probable that a uniform feed rate would increase cell capacity, reduce coal loss in the refuse, and reduce reagent consumption.

Flotation of Raw Coal, Kimberly Plant

The feed to the Bessie flotation plant is a sludge or fine material which has already been washed in the jig plant and from which the extreme fines have been removed by classifying action in the settling tank. Experimental work by Gandrud[1] had shown that a raw coal feed could be handled as effectively as sludge except that conditions for dewatering were more exacting. There are a number of advantages to be gained by floating raw coal, such as:

1. All the fine coal, including the finest sizes, are retained as cell feed so that coal otherwise lost in the effluent from the plant may be recovered. Based on determinations of solids in the effluent, this loss at Bessie is estimated at 20 tons per day.
2. Added washer capacity may result. The removal of fines from the equipment for washing the coarser sizes may allow this equipment to operate at greater capacity, thus increasing the overall washing capacity of the plant.
3. The removal of the fines may permit better jig performance.
4. The removal of the fines as raw coal avoids the need for settling tanks and attendant slurry problems.

In view of these possible advantages, it was decided the next installation should be arranged for raw coal feed. One point on which there was doubt was the matter of screening the raw coal to make a 10-mesh separation. As it arrives at the tipple, coal contains too much moisture for dry screening and, if wet screening were done in the usually accepted manner for wet screening, which calls for 5 to 10 gpm per cu yd per hr of screen feed, the resulting underflow pulp would be more dilute than desired. Screen manufacturers willingly furnished information on capacities for either wet or dry

Table II. Typical Screen Analyses, Bessie Mine Flotation Plant (Cumulative Pct)

	Pct Retained On,							
	6 Mesh	10 Mesh	14 Mesh	20 Mesh	35 Mesh	65 Mesh	100 Mesh	200 Mesh
Cell feed	6.1	18.2	29.1	45.2	72.8	86.5	91.3	95.5
Floated coal								
No. 1 cells	0	4.1	14.0	31.8	70.3	85.1	90.5	
No. 2 cells	1.1	8.1	20.9	41.8	81.4	91.1	91.7	
No. 3 cells	0.4	8.4	16.0	27.8	76.1	86.7	88.6	
No. 4 cells	1.1	13.8	29.9	50.5	87.6	93.3	94.1	
No. 5 cells	0.7	16.3	30.5	50.0	80.5	91.2	92.2	
No. 6 cells	2.1	20.1	35.6	55.2	77.7	82.5	84.6	
Total, all cells	0.3	6.2	17.2	36.9	71.4	86.2	91.4	95.9
Secondary coal (0' size)	22.6	50.4	67.9	85.0	98.7	99.6		
Refuse	0.5	2.1	4.3	10.8	40.5	63.8	73.5	84.9

screening, but there seemed to be a lack of data available for conditions intermediate between wet and dry. For lack of better information, it was decided to determine screen capacity on the basis that the amount of water required for the desired pulp density would give the coal sufficient mobility to make the screening capacity equivalent to that of dry screening. If it was found that more water was needed, part of it would be recirculated.

Kimberly mine in the Jefferson seam was selected as the logical place for experimenting with a full scale plant on raw coal feed for the following reasons:

1. Kimberly is a comparatively small mine having an output of 450 to 500 tons per day so that for an initial trial of this method, the installation could be made at the least expense.

2. Additional washing capacity may be needed at Kimberly because plans are being made to increase the tonnage. It was hoped the cell plant would increase washer capacity.

3. The jig at Kimberly is a Montgomery jig, a type of jig with a fixed screen under which a plunger with flap valves provides the jigging strokes and pumps the water required. Tests of the washed product indicated that jig separation became poor at —⅛ in. as compared with —35-mesh for the Baum type jig.

4. The fines in the Kimberly washed coal were found to be quite high in ash so that if raw coal flotation was successful, an overall reduction of 3 pct could be expected in ash content of the washed coal from this mine. Such reduction would give a greater return per dollar of invested capital than could be obtained at the other mines.

On the basis of experience at Bessie, it was decided a single row of six 100-cu ft No. 30 (56x56) Denver "Sub-A" cells would have a capacity of 20 tph, which was expected eventually to be the required capacity at Kimberly. The plant was laid out and built with this arrangement and went into service in April 1949.

To understand the operation at Kimberly, a brief flowsheet description is necessary. Coal crushed to 1½ in. at an underground screening and crushing station is brought from the mine by conveyor belt to a 300-ton storage bin. From this bin a feeder and 24-in. belt bring the coal to the washer where it is discharged on a single-deck, 5x14-ft vibrating screen equipped with a No. 5185 ton cap screen cloth having an equivalent opening of approximately 9-mesh. Three rows of sprays cover the deck with sprayed water at a rate of 250 gpm. The oversize from the screen discharges into a 50-tph Montgomery jig. The washed coal from the jig is conveyed by bucket elevator and drag to the loading bin, the refuse by bucket elevator to the rock bin from which it is hauled to waste by truck.

The underflow from the vibrating screen falls into a hopper from which an 8-in. pipe conveys it to the flotation cells. There are six cells in series, the pulp making a single pass through, with unfloated material being rejected at the last cell. Reagent flows by gravity from a 6000-gal storage tank through a cup feeder and distributor to the hopper under the screen (75 pct) and to the inlet of the fourth cell (25 pct). The floated coal is removed by a raking mechanism in each cell and discharged into a 14-in. dewatering screw. Near the discharge end of the screw are two sections of 1-mm wedgewire screen 60 in. long, 11 in. wide and curved to 7-in. radius through which water is squeezed by the screw action. The water so removed is returned by a return feed opening into the impeller zone in the first cell. The screw uses a 14-in. pipe as a trough, and this pipe projects 20 in. beyond the end of the screw, forming a nose through which the screw extrudes the coal, thus affording a squeezing action which aids in water removal. It is to the bottom of this pipe that the curved wedgewire screen is attached. The floated coal discharges from the screw into a drag conveyor which carries it up a 35° incline to the coarse coal drag where it mixes with the main body of coal and proceeds to the loading bin. The unfloated material leaving the last cell is discharged into a 5-ft diam x 5-ft high dewatering cone. The underflow from this cone containing the coarser part of the rejected material is pumped by a 2-in. sand pump to a 3x6-ft vibrating screen equipped with 10-mesh cloth. Screen oversize joins the coarser coal going to the jig. The screen undersize joins the overflow from the dewatering cone, the two together constituting the refuse from the flotation plant. It will be noted that this arrangement of dewatering cone and vibrating screen for unfloated material is similar to that at Bessie. This screening arrangement has a number

Table III. Operating Costs of Flotation Plants Per Ton of Feed

BESSIE

Operating labor	$0.00
Reagent, 0.75 gal kerosine at $0.10 per gal	0.075
Power, 2.9 kw-hr at $0.01 per kw-hr	0.029
Maintenance and supplies, (est.)	0.03
Total cell plant proper	0.134
Water supply, 250 gpm against 125 ft head	0.004
Total operating cost per ton of feed	$0.138

KIMBERLY

Operating labor	$0.00
Reagent, 0.7 gal kerosine at $0.10 per gal	0.07
Power, 4.5 kw-hr at $0.015 per kw-hr	0.068
Maintenance and supplies, (est.)	0.04
Total cell plant proper	0.178
Power for water supply screening and material transport, 2.8 kw-hr at $0.015 per kw-hr	0.042
Maintenance water supply, screening and material transport, (est.)	0.02
Total operating cost per ton of feed	$0.240

Table IV. Kimberly Operating Statistics, Average of Daily Results

Seam, Jefferson; output 475 tons coking coal per day

	With Flotation Month of August 1949	Prior to Flotation 6 Months Ending March 31, 1949
General Washer Results		
Raw coal 1½x0, ash	22.4	22.5
Washed coal ½x0, ash	6.7[a]	10.4
Jig refuse, ash	70.4	66.0
Float in jig refuse at 1.45 sp gr, pct	3.8	4.8
Hutch solids, ash	51.0[b]	36.3[b]
Flotation Details		
Raw coal to cells, 10 mesh x 0, ash	23.5[b]	
Floated coal, ash	5.1	
Cell reject oversize to jig, ash	31.6[b]	
Refuse from flotation, ash	58.2	
Reagent 4.5 lb kerosine per ton of feed		

[a] Includes floated coal.
[b] Average of a few snap samples. Routine samples not available.

Table V. Typical Screen Analyses, Kimberly Mine Flotation Plant (Cumulative Pct)

	Pct Retained On							
	6 Mesh	10 Mesh	14 Mesh	20 Mesh	35 Mesh	48 Mesh	100 Mesh	200 Mesh
Cell feed	0.1	5.1	14.6	28.3	52.8	66.9	81.1	91.0
Floated coal								
No. 1 cell	0	1.0	5.4	15.8	44.4	58.9		
No. 2 cell	0	1.6	8.1	21.4	50.8	63.3		
No. 3 cell	0	2.4	9.8	25.8	54.7	66.8		
No. 4 cell	0	2.6	10.7	26.5	55.4	67.7		
No. 5 cell	0	3.1	13.2	29.2	56.0	66.9		
No. 6 cell	0	5.2	18.2	36.0	60.4	69.1		
Total, all cells	0	2.6	11.7	26.0	52.9	64.3	79.9	89.3
Oversize to jig	2.1	52.4	86.4	96.2	99.0			
Refuse	0.1	4.1	13.6	26.1	47.6	55.1		

of functions. First, and primarily, it serves as a guard for the circuit, protecting against loss of coal when oversize material too large for flotation appears in the cell circuit. Second, it prevents the loss of intermediate gravity coal near the upper limit of flotation size. While kerosine flotation will float particles of 10-mesh or larger of low gravity, intermediate gravity material does not float as readily and 10-mesh particles of coal of rather high ash, but too good to throw away, would otherwise be lost in the refuse. Third, the use of the guard screen permits using a mesh size on the main screen that is the top size for flotation, thus rendering the main screening job easier. Incidentally, this screen provides a very good method of checking the flotation circuit at a glance. Normally there is only a dribble of oversize on the screen. When the amount of oversize on the screen becomes appreciable, something is wrong with the flotation circuit or the feed to the circuit.

Under present operating conditions, the flotation plant receives a feed of about 12 tph of —10-mesh coal of about 23 pct ash at a pulp density of between 15 and 20 pct. It delivers about 7¾ tons of floated coal assaying 5 pct ash, returns ½ ton of unfloated material to the jig at about 30 pct ash, and sends 3¾ tons of refuse containing 58 pct ash to waste.

Table IV gives a summary of operating results at Kimberly mine for the month of August 1949, and table V gives typical screen analyses of cell feed and products.

Flotation of raw coal feed has proved quite successful and the advantages that were expected to result from this method have been realized.

1. Retention of fine coal. Wasting the hutch discharge is the method used at Kimberly to bleed water from the circulating system. Prior to the installation of the flotation units, the hutch discharge from the jig assayed 36 pct ash, indicating a loss of coal roughly estimated at 6 tons per day. The hutch material is now about one tenth the former quantity with an ash content of about 51 pct, representing a negligible loss.

2. Greater capacity on the jig. After installation of flotation, the elevator conveying the washed jig product was speeded up approximately 20 pct and a further increase in speed is contemplated.

3. Better jig performance. Prior to flotation the ash in the jig refuse averaged 66 pct and the float at 1.45 sp gr in the jig refuse, 4.8 pct. Now the jig refuse ash is 70 pct and the float in the refuse 3.8 pct.

The average ash of the Kimberly output for 6 months prior to cell installation was 10.4, the average ash for the first 5 months of cell operation was 7.0 pct.

The screening performance has exceeded expectations. Kimberly coal is extremely friable and efficiencies of screening are therefore hard to determine. If calculated on the fines available in the feed and the fines appearing in the undersize, the efficiency reported is too high. If calculated by combining the undersize and oversize and determining the percentage the undersize is of this total, the reported efficiency should be nearer its true value. The latter method is used in obtaining the figures given here. Two tests were made of screen efficiency. Both tests were made at rates in excess of the usual feeding rate of 1.12 tph per sq ft of screening surface because they were made in an effort to determine screen capacity. The usual spraying rate is 2.9 gpm per cu yd of feed per hr.

At a feed rate of 1.52 tons dry coal per hr per sq ft and a spraying rate of 3.0 gpm per cu yd per hr, 82.5 pct of the 10-mesh x 0 appeared in the undersize, 89 pct of the 14-mesh x 0 and 93 pct of the 20-mesh x 0.

At a feed rate of 1.80 tons per sq ft per hr and a spraying rate of 2.6 gpm per cu yd per hr, 77 pct of the 10-mesh x 0 appeared in the undersize, 83 pct of the 14-mesh x 0, and 87 pct of the 20-mesh x 0. Table VI gives screening date on these two tests.

As in the case of Bessie, no accurate means for regular determination of the quantity of feed to the flotation plant is available so that cost data is based on an occasional determination of feed rate and is therefore approximate. Maintenance is approximated on the basis that cell-wearing parts are replaced every two years, plus an amount for other maintenance based on Bessie experience. Cost of operation per ton of feed is given in table III. Prior to cell installation, one man operated the washer. With the addition of the cell units, one man still operates the washer, attending to both the jig and the flotation unit so that no labor cost is shown for flotation.

Cell Capacity

The capacity of a flotation unit depends on many factors among which are pulp density and the permissible loss of values in the rejected material. In the Sloss units, the pulp density decreases from cell to cell as floated coal is removed, and the amount of coal floated per cell also decreases. To avoid appreciable loss of coal in the refuse, experience indicates that the amount of coal floated in the last cell should not exceed 1 tph for a 100 cu ft cell or ½ tph for a 50-cu ft cell.

The distribution of floated coal by cells varies with loading and other factors. Two typical examples of distribution with normal loading as determined by test are given here.

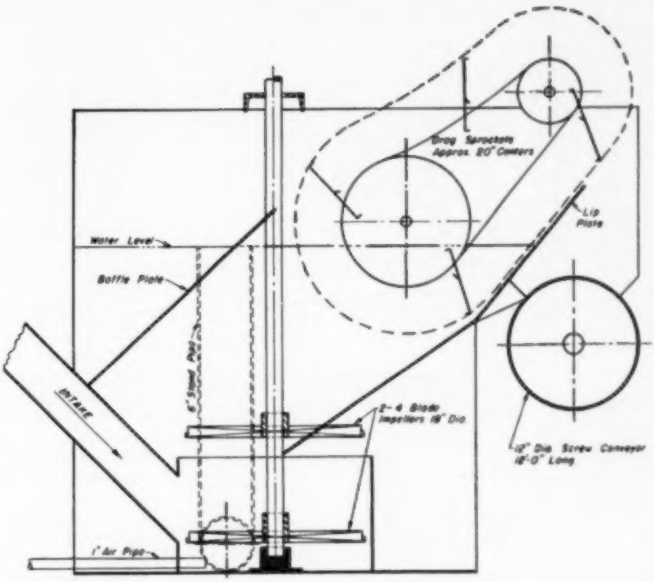

Fig. 4a—Flocculation cell for the flotation of sludge coal. Bessie mine.

At Bessie, with the float from the first cell taken as unity, the float from the second cell was found to be 0.80; from the third cell, 0.37; from the fourth cell, 0.33; from the fifth cell, 0.15; and from the sixth cell, 0.11. Likewise at Kimberly, with the float from the first cell as unity, the float from the second was found to be 0.98; from the third, 0.96; from the fourth, 0.78; from the fifth, 0.50; and from the sixth, 0.40.

The Bessie installation was intended to handle an average of 22 tons of dry feed per hour and is doing approximately that at present. Since a secondary coal product is being made from the cell reject, its capacity is probably greater than it would be without the secondary product unless a lower ash refuse were accepted.

The Kimberly installation was designed to handle 20 tons of dry feed per hour and is at present handling about 12 tons with very satisfactory results. Capacity tests of brief duration have been made at feed rates of approximately 20 tph with disappointing results. The floated product was satisfactory, but too much coal reported in the unfloated material, the refuse ash being about 45 pct. The top satisfactory capacity is therefore somewhere between 12 and 20 tph of feed. Until further information is available, a conservative capacity rating is 12 tph.

The approximate cost of the Bessie installation in 1947 was $32,000, 87 pct of which can be apportioned to flotation equipment and auxiliaries and 13 pct to housing. No added facilities were required for water supply or transport of material to or from the flotation plant. At current costs, a similar plant now would be estimated at $40,000, or $1800 per ton-hour of capacity, when rated at 22 tph of feed.

The approximate cost of the Kimberly installation, which was completed in 1949, was $35,000. This includes an added water supply, screening facilities, and transport of finished product from the plant. It can be apportioned 61 pct for flotation equipment and auxiliaries; 26 pct for water, screening and transport; and 13 pct for housing. At a conservative rating of 12 tph of feed, this is approximately $2900 per ton-hour of capacity or about $2200 for the cell plant alone.

Investigation of Methods for Increasing Capacity

While both the Bessie and Kimberly plants are of sufficient capacity to handle the available feed, other installations are contemplated, and therefore it has

Table VI. Screen Analyses of Feed to and Products from 5x14 Robbins Vibrex Screen, Kimberly Mine (Cumulative Pct)

	¼ In. Round	6 Mesh	10 Mesh	14 Mesh	20 Mesh	35 Mesh	65 Mesh	100 Mesh	200 Mesh
Test No. 1									
Feed	50.0	64.4	77.4	83.2	87.3	93.8	96.5	97.7	99.1
Oversize (to jig)	81.5	90.0	95.0	97.5	98.8	99.7			
Undersize (to cells)	0	0	8.0	20.5	36.5	61.3	77.6	84.8	92.4
Test No. 2									
Feed	52.3	67.1	79.4	84.8	88.8	94.2	96.6	97.8	99.2
Oversize (to jig)	78.3	87.1	93.7	96.2	97.6	99.3			
Undersize (to cells)	0	0	6.0	17.6	32.8	60.0	76.4	83.5	92.5

Screen cloth, ton cap No. 5185 — Screening area, 62 sq ft

Test No. 1 — 1.52 tons dry feed per sq ft screen area per hr; 3.0 gpm water sprayed per cu yd feed per hr

Test No. 2 — 1.80 tons dry feed per sq ft screen area per hr; 2.6 gpm water sprayed per cu yd feed per hr

Feed — 1½ in. x 0 raw coal. Weight, 60 lb per cu ft

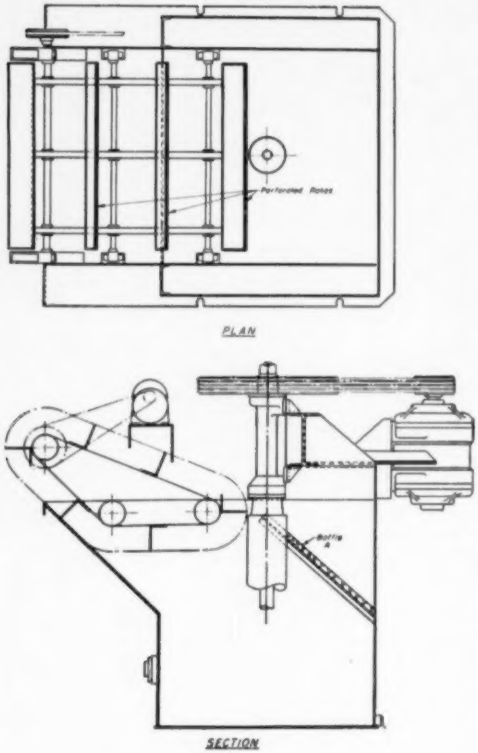

Fig. 4b—Standard raking mechanism.

been desirable to investigate means of increasing cell capacity for the benefit of these future installations. Investigations have been made on the following possible methods of securing such increase: supercharging, changing impeller speed, increasing area covered by raking, and conditioning.

Super Charging: When the first 100-cu ft cells were purchased, the Denver Equipment Co. which supplied them recommended supercharging be used to develop the full capacity of the larger cells. Accordingly, the Bessie installation was arranged for supercharging of both the 100 and 50-ft cells. No advantage was found in the use of supercharging and, after a few days of experimenting, supercharging was discontinued. After the Kimberly installation was made, observation of some turbulence breaking through the surface of the matte of floated coal led to the belief that a reduction in air might be advantageous. Experiments were made with the second cell by gradually closing the air intake. As a final result, the intake was completely closed with a resulting 7 pct increase in the amount of coal floated on this cell. No attempts have been made as yet to verify this on other cells, and the data given in this paper was secured with all intakes open.

Changing Impeller Speed: Observation of the same turbulence previously mentioned suggested a lesser impeller speed might be advantageous. Accordingly, arrangements were made at Kimberly to slow the impellers in the first and second cells from a speed of 253 rpm to a speed of 207 rpm. Some quieting of turbulence was noted, the power input to these two cells was halved, but the first cell floated only about 80 pct of its normal amount and the second cell 90 pct of normal. Although the remaining cells, with more floatable coal available, floated somewhat more than their normal amount, the ash in the cell refuse averaged 48 pct compared to a normal 58 pct. Since a decrease in speed of 18 pct for the first two cells results in a decrease in float capacity averaging roughly 15 pct, conversely, an increase in speed can be expected to develop an increase in capacity, but such increase would be accompanied by a rapid rise in power consumption and probably by a rapid increase in wear of moving parts.

Increasing Area Covered by Raking: When the Kimberly installation was being planned, Denver Equipment Co. offered as an alternate arrangement a cell equipped with a spitzkasten and raking mechanism on each side instead of the cell with the single spitzkasten and raking mechanism as furnished for Bessie. No data was available, but such an arrangement might offer a considerable capacity increase. Installation of double raking was thoroughly discussed, but space limitations and the complication of handling material from two dewatering screws swung the decision in favor of the Bessie type of cell. After Kimberly went into service, discussion of the advantages of double raking over single raking continued. At length, the idea evolved that even with the single spitzkasten and single dewatering screw, the raking mechanism might be rearranged to reap the advantages of double raking. The second cell at Kimberly was picked to make the experiment. The standard raking mechanism is shown in fig. 4*b*. The mechanism as rearranged in the second cell is shown in fig. 4*c*. The change involved the removal of baffle *A* in fig. 4*b*. The Experimental arrangement rakes an active cell area 2.3 times as large as the standard mechanism does; or, if the spitzkasten area is included, 1.7 times as large. With standard rakes, the second cell usually discharges 0.98 tons of floated coal for each ton discharged by the first cell. With the experimental mechanism, the second cell discharges an average of 1.05 tons of floated coal for each ton by the first cell, indicating an increase in capacity of about 7 pct.

The experimental arrangement rakes as much active cell area as would the double raking mechanism and is believed to possess substantially the same advantages without the extra space required by the second spitzkasten or the need for a second dewatering screw.

Conditioning: The original experimental homemade flotation unit at Bessie was preceded by a rather crude homemade conditioning unit for agitation of the mixture of pulp and reagent. In the experimental work, no evidence was found of advantage resulting from the use of the conditioning unit, so conditioning was omitted from the Bessie installation and later from the Kimberly installation. When a search was begun for means of increasing cell capacity, the idea of conditioning was revived. It should be noted that, when testing slower impeller speed at Kimberly, the float capacity loss in the second cell was about one half that in the first, and it is therefore probable that conditioning ahead of flotation would permit slower impeller speeds with little or no loss in capacity. Slower

speeds would decrease both power and maintenance costs. It was not convenient to add experimental conditioning equipment to either the Bessie or the Kimberly installation so investigation was confined to a few laboratory tests. The tests were made with a 500-g Denver laboratory cell. Without conditioning, 90 pct of the float came off in 6 min. With 5-min conditioning, 90 pct came off in 5 min; and with 10-min conditioning, 90 pct came off in 4.2 min. Since in the Bessie installation the major portion of the coal is floated in the first two cells and since calculations show a retention time of roughly 5 min in the larger cells, making a conditioning period of about 10 min before entrance of the pulp to the third cell, the use of a conditioner has not seemed necessary at Bessie. Nevertheless, in view of the laboratory results, it is probable that conditioning would result in increased capacity. It is also probable that conditioning would decrease reagent consumption although no laboratory tests have been made to determine this.

Although they have not been investigated, two other methods of increasing capacity have been considered, namely, increasing initial pulp density to the cells and increasing pulp density somewhere along the row of cells by removal of water.

Gandrud has shown by laboratory work that the cell capacity increases as pulp density increases until a given density is reached after which capacity falls off rather abruptly. In the laboratory the optimum density found was 28 pct solids. The optimum for commercial cells might be decidedly different. In order to gain capacity by operating at as near optimum density as possible, it undoubtedly would be necessary to have some means of securing a uniform rate of feed, which is lacking at Bessie and Kimberly.

Increasing pulp density somewhere along the row of cells could be done by a number of methods, such as the use of a dewatering cone inserted between cells to eliminate part of the water or a classifier for the same purpose. In an installation of the size discussed in this paper, the methods so far proposed involve complications that make it cheaper to obtain capacity by the purchase of additional cells. For a large installation such methods may have merit.

Reagent

The reagent used at both Bessie and Kimberly at the present time is kerosine with no additions. All of the original experimental work was done with a mixture of about 93 pct kerosine and 7 pct B23 frother as reagent. The Bessie installation was started using this same mixture. The B23 costs about $1.05 per gal as compared with about $0.10 for kerosine so that its use adds materially to the reagent cost unless accompanied by a marked reduction in reagent requirements. In order to determine the value of the B23 addition, the plant was operated for several alternate weeks with and without the addition. Although laboratory work has repeatedly shown the superior performance resulting from the use of the frother, no definite advantage could be proved by the actual plant performance. The use of B23 was discontinued.

After the Kimberly plant was in operation, several brief snap sample tests under the supervision of Gandrud and Riley[2] were made using both pine oil and B23 as an addition. These tests gave improved results when pine oil was added and still greater improvement when B23 was added. Except for these brief test interludes, only straight kerosine without additions has been used as reagent at Kimberly. With the present rate of feed, the cell operating results have been considered satisfactory. Should an attempt be made to increase capacity, the use of additions to the kerosine probably would be considered again.

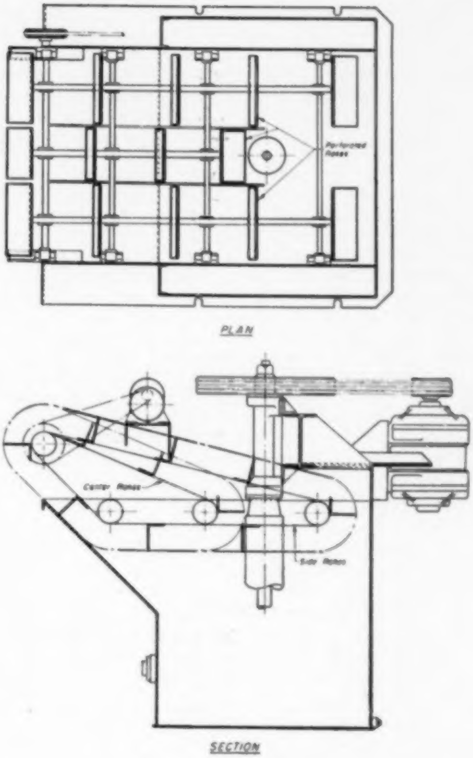

Fig. 4c—Raking mechanism as rearranged in the second cell, Kimberly.

Dewatering

One of the advantages of the kerosine flotation process is that the floated coal can be readily dewatered to about 25 pct moisture by the use of a very simple device, the dewatering screw. This device is described in detail by Gandrud and Riley.[1] In the Sloss installations, this advantage has not been fully developed because moisture reduction to this extent has not been needed. At Bessie the moisture content of the coal as discharged from the screw varies from 34 to 45 pct with an average of about 37 pct; at Kimberly, from 44 to 52 pct with an average of about 47 pct.

The floated coal drains readily without loss of coal because the process has put it in a flocculated or matte condition, and because most of the fireclay usually in the fines has been removed in the process. For instance, a sample of Kimberly floated coal that was discharged from the screw with 51 pct moisture

drained to 39 pct in ½ hr, 35 pct in 3 hr, and 32 pct in 22 hr. Thus the floated coal unmixed with coarser material will drain in a day to a moisture content approaching that of an effective dewatering screw. When mixed with coarser coal, as is the case at both Bessie and Kimberly, it drains readily in the loading bin and railroad cars so that upon arrival at the coking plant, 24 to 48 hr later, the moisture content has declined to a point where little would be gained by more effective initial dewatering. For example, Bessie coal, prior to the installation of flotation equipment, averaged 7.8 pct moisture as delivered to the coking plant and, since installation, averages 7.9 pct. Kimberly, prior to installation of flotation, averaged 6.5 pct and, since installation, averages 6.3 pct. Flat Top mine, which has no flotation units, averages 8.3 pct and Lewisburg mine, which also has no units, averages 8.9 pct.

At Bessie the floated coal mixed with coarser coal is conveyed by drag conveyor up a 34° incline, and at Kimberly the floated coal alone by drag conveyor up a 35° incline. As long as the coal is sufficiently dry to be conveyed successfully up these inclines, there is no need for further dewatering. Consequently, the screen area for draining purposes in the dewatering screw has been held to a minimum, and no attempt has been made to develop the full capability of the screw.

Efficiency

The effectiveness of the process in making a proper separation during regular operation is judged by routine ash analyses of the floated coal and refuse and by observation of the amount of oversize reject going to the jig (Kimberly) or into secondary coal (Bessie). No tests of efficiency are available for Bessie. For Kimberly, an investigation has been made by Gandrud and Riley[2] when using kerosine and pine oil as reagents, and the results are given in a Bureau of Mines Report of Investigation about to be published from which the following excerpts are taken:

On the basis of the "error curve" method for evaluating performance, the plant in which the raw coal fines are treated compares favorably with plants using the more conventional type of processes such as jigging, launder washing and froth flotation.

The head sample for the float-and-sink tests . . . was obtained by mixing —10-mesh floated coal and —10-mesh refuse from the flotation unit in proportionate amounts on the basis of recovery weights. This had to be done in order to eliminate the conditions resulting from size degradation of the feed during treatment in the cells. This degradation on Kimberly coal is so severe that error curves . . . could not be plotted except on the basis of float-and-sink data representing feed after degradation had taken place.

. . . all material coarser than 10-mesh was screened out of the samples and excluded from the data. . . . The feed rate of the flotation unit was 11.5 tons per hour. The reagent feed was at the rate of 3.04 pounds of kerosene and 0.26 pounds of pine oil per ton of feed. Of this, about 65 pct of the kerosene was added to the feed in the intake pipe of No. 1 cell. The remainder of the kerosene and all of the pine oil were added to the weir compartment ahead of No. 4 cell, in other words, in the intake of No. 4 cell. The amount of water in proportion to solids could not be determined accurately but was probably about 5 to 1, resulting in about 17 pct solids in the feed.

The figures . . . show that the composite 10-mesh to 0 products analyzed 21.5, 6.3, and 63.6 pct ash for the feed, cleaned coal and refuse, respectively, and that 73.5 pct of the feed was recovered as cleaned coal. According to the cumulative curve . . . the float-and-sink yield at 6.3 pct ash was 76.7 pct. Hence the efficiency of the flotation treatment was 73.5 divided by 76.7 or 95.8 pct. This efficiency may seem a little low in comparison with efficiencies normally attained in jig and table washing on the coarser sizes, but, as a matter of fact, the situation would have been considerably different on the basis of the 10-mesh to 0 feed prior to treatment in the cells. Float-and-sink tests made on a sample of 10-mesh to 0 feed taken ahead of the flotation cells showed only 74.3 pct yield at 6.3 pct ash instead of 76.7 pct shown . . . for the composite sample which had undergone size degradation in the cells as explained above. On this basis, the recovery efficiency would have been 73.5 divided by 74.3 or 98.9 pct.

In kerosine flotation, as in coarser coal cleaning methods, the effectiveness of separation becomes poorer below some limiting size. In the case of kerosine flotation, the size at which separation becomes considerably poorer appears to be 200-mesh or below. On a Kimberly floated coal sample of 6.1 pct ash content, the ash in the +48-mesh was 6.0 pct; in the 48x65-mesh, 5.0 pct; in the 65x100-mesh, 5.1 pct; in the 100x150-mesh, 5.7 pct; on the 150x200-mesh, 6.2 pct; and in the —200, 8.6 pct.

The probable cause of the increase in ash in the —200-mesh particles is entrapment of fine refuse and suspension of fireclay in the water carried with the floated coal from the cells.

Summary

Seeking an answer to the problem of cleaning fine bituminous coal, the Sloss-Sheffield Steel and Iron Co. collaborating with B. W. Gandrud, U. S. Bureau of Mines, has experimented on a successively increasing scale with the kerosine flotation process for coal fines. The experimental work has led to the construction of two commercial flotation plants which are now in operation.

The first plant, with a feed of roughly 22 tph of 20 pct ash —10-mesh sludge from a jigging operation, is producing a floated product of about 8 pct ash content at an operating cost of $0.14 per ton of feed with a capital investment of approximately $1800 per ton-hour of capacity.

The second plant, with a feed of roughly 12 tph of 23 pct ash —10-mesh raw coal screened out ahead of the jigging operation, is producing a floated product of about 5 pct ash content at an operating cost of $0.24 per ton of feed with a capital investment of approximately $2300 per ton-hour of capacity or $2900 per ton-hour for the complete installation.

Further investigation and experiment should lead to an increase in the floated coal obtained per cell unit with a resulting decrease in the capital investment required per ton-hour of capacity and also a resulting decrease in the operating cost per ton of feed.

References

[1] B. W. Gandrud, and H. L. Riley: A Combination Cleaning and Dewatering Process for Treating Fine Sizes of Coal Preliminary Report. U. S. Bur. Mines. *R. I.* 4306.

[2] B. W. Gandrud, and H. L. Riley: Recent Developments in Connection with a Combination Cleaning and Dewatering Process for Treating Fine Sizes of Coal. U. S. Bur. Mines. *R. I.* In press.

Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal

by M. R. Geer, H. F. Yancey, C. L. Allyn, and R. H. Eckhouse

Four coals were treated in the Humphreys spiral concentrator, and the products were examined by float-and-sink and screen-sizing tests to determine fundamental performance characteristics. The efficiency of the separation between coal and impurity was shown to be influenced greatly by particle size.

Summary

FOUR coals exhibiting different washability characteristics were washed in the Humphreys spiral concentrator and the products examined by float-and-sink and screen-sizing tests to determine the mechanics of the separation between coal and impurity. With one coal, each ⅓-in. increment of the full width of the stream in the spiral was examined separately to determine where each size and specific-gravity fraction of the feed was stratified.

The basic performance characteristics of the spiral were found to be the same for all of these coals. The coarsest fraction of heavy impurity stratified so far out in the stream that it could not be removed through the refuse ports and thus either entered a middling product or contaminated the washed coal in the case of a two-product separation. Impurity particles finer than about 100-mesh also were carried out in the main body of the stream and therefore were not removed in the refuse product. Little loss of clean coal in the refuse occurred in sizes coarser than 28-mesh or finer than 100-mesh, but considerable coal of intermediate size stratified in the stream that was drawn off through the refuse ports.

Because of this modifying influence of particle size, the spiral is unable to make an efficient two-product separation between coal and impurity without some retreatment. Recirculation of a middling product through the same spiral as a means of obtaining more efficient operation was not attempted in this investigation. The size and specific-gravity composition of the middling product are, however, such as to render it amenable to simple hydraulic classification for recovery of coal. Retreatment of a middling product would improve the efficiency of the spiral, but even when making a substantial amount of middling, considerable clean coal enters the refuse product. Consequently, retreatment of a combined refuse-middling product would appear to offer greater promise for providing maximum efficiency.

Because of the sizing characteristics of the spiral, a classified feed can be treated with higher efficiency than is possible on a natural raw coal. For this reason, semiclassified feeds, such as silt-bank or classifier-underflow materials, can doubtless be treated with higher efficiencies than those shown for raw coals in this report.

Performance is, of course, only one of many factors that enter into the choice of cleaning equipment. The fact that the one company now using the spiral for cleaning coal has built a second spiral plant is ample evidence that the inherently low efficiency of this unit may be overshadowed by the low operating costs afforded by its extreme simplicity.

Introduction

A number of factors enter into the selection of coal-cleaning equipment, and most of them bear directly on the final cost of prepared coal. One of these factors is performance of the cleaning unit in terms of the efficiency of the separation it effects between clean coal and refuse. The importance of performance in the selection of a coal-cleaning unit varies with the difficulty of the cleaning problem. Obviously, if the character of the coal is such that only certain types of cleaning units are capable of yielding a clean product meeting market requirements, performance is of prime importance. If, on the other hand, the cleaning problem is so simple that any type of cleaner will provide coal of suitable quality, performance is important only insofar as it affects costs by determining the amount of salable coal lost in the washery refuse.

With the objective of providing the coal industry with information that is helpful in the selection of

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Fig. 1—Full-size spiral installed in laboratory.

coal-cleaning equipment, the Bureau of Mines has published the results of detailed performance studies of the wet table,[1] pneumatic table,[2] Dutch cyclone,[3] and jigs of pneumatic,[4] plunger,[5] pulsator,[6] and Baum[7] types. The present report contains such information for the Humphreys spiral concentrator.

This spiral was developed during the recent war to treat the chrome-bearing sands of Coos Bay, Oregon. In the few years that have elapsed since its development, the spiral has been adopted for treating ores, nonmetallics, and finally coal. One coal-washing plant utilizing the spiral went into operation in the anthracite field in 1946,[8] and the same company recently has completed a second spiral plant. In addition, experiments on a pilot-plant scale were carried out with the spiral at a washery in Colorado.[9]

The spiral is an extremely simple device which involves no moving parts and is constructed almost entirely of unmachined castings. Since it is such an uncomplicated mechanism, operation is simple and virtually foolproof. These characteristics, which go far toward insuring low-cost operation, are attractive attributes in any coal-cleaning unit. Moreover, published information[8] indicates that the spiral can treat coal containing 30 to 40 pct ash to produce a washed product of 14 to 16 pct ash, with rejection of a refuse product containing up to 80 pct ash. Thus, the spiral was promising enough to warrant a detailed performance study.

Object and Scope

The three published reports describing operation of the spiral contain some performance data; but, in the case of the anthracite plant, this information is not sufficiently complete to provide a clear picture of the mechanics of spiral performance. More detailed information was published for the pilot-plant operation in Colorado; but this unit was used exclusively for treating a table-middling product experimentally, and thus the data are not directly applicable to the treatment of raw coal.

The object of the present investigation was to provide detailed information on the influence of particle size and specific gravity in the separation between clean coal and refuse effected by the spiral. With any gravity concentrating device, except some heavy-medium processes, particle size modifies the influence of specific gravity. The magnitude and character of the influence of particle size determine the fundamental performance characteristics of any cleaning device.

A laboratory installation is ideally suited for determining basic performance characteristics because operating variables can be controlled. On the other hand, a laboratory installation provides little or no information on the other factors bearing on cost that enter into the selection of cleaning equipment. Consequently, the present report is limited in scope to performance data only and necessarily neglects the other factors that can be established best in plant operation.

Nevertheless, this investigation was intensive—over 60 test runs were made with the spiral, and the products obtained from most of them were examined by float-and-sink tests and screen analyses, rather than just ash analyses. Therefore, the conclusions reached regarding fundamental spiral performance characteristics are based on abundant detailed data for four different coals and hence are generally applicable.

Description of Coals Tested

Coals from Alabama, Kentucky, West Virginia, and Washington were used in this investigation. Table I gives the source of these coals, and tables II, III, IV, and V give float-and-sink and screen-sizing data. All samples represent natural 8-mesh fines screened from run-of-mine coal, except the West Virginia, which was prepared by crushing 3-in. slack to pass 8-mesh.

Table I. Identification of Coals Tested

State	County	Bed	Mine
West Virginia	Wyoming	Pocahontas No. 6	Black Eagle
Alabama	Jefferson	Clements	Prospect
Kentucky	Union	No. 9	Poplar Ridge
Washington	Kittitas	Roslyn	Roslyn No. 3

The Black Eagle coal from West Virginia contains only 5.3 pct of impurity heavier than 1.60 sp gr. With this coal, the coarsest and finest sizes are about equally dirty, and the least impurity is found in the material of intermediate size.

The Clements coal from Alabama contains slightly more heavy impurity than that found in Black Eagle but has very little material of intermediate specific gravity, the presence of which is generally regarded as an index of how difficult a coal is to wash. The impurity in this coal tends to be concentrated in the finer sizes.

The Poplar Ridge coal from western Kentucky is much dirtier than either the Black Eagle or Clements coals, containing nearly 17 pct of impurity heavier than 1.60 sp gr. This coal contains the highest percentage of material of intermediate density of any of those tested. Also, the heavy impurity contaminating it is predominantly fine material; the coal finer than 200-mesh contains over 40 pct of heavy impurity.

The Roslyn coal from the State of Washington contains slightly more heavy impurity than that found in Poplar Ridge coal but somewhat less material of intermediate density. With Roslyn coal there

Table II. Specific Gravity Analyses, by Size Fractions, of Black Eagle, W. Va., Coal

Size, Mesh	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
8 to 14	Under 1.30	26.6	1.9	26.6	1.9
Weight, 26.9 pct	1.30 to 1.40	56.7	5.5	83.3	4.4
	1.40 to 1.60	10.6	17.7	93.9	5.9
	1.60 to 1.80	1.6	38.2	95.5	6.4
	Over 1.80	4.5	81.9	100.0	9.8
14 to 28	Under 1.30	38.0	1.5	38.0	1.5
Weight, 28.1 pct	1.30 to 1.40	48.5	5.3	86.5	3.6
	1.40 to 1.60	9.0	17.2	95.5	4.9
	1.60 to 1.80	1.2	36.3	96.7	5.3
	Over 1.80	3.3	80.0	100.0	7.8
28 to 48	Under 1.30	41.2	1.4	41.2	1.4
Weight, 19.6 pct	1.30 to 1.40	46.1	4.9	87.3	3.2
	1.40 to 1.60	8.6	16.7	95.9	4.5
	1.60 to 1.80	1.1	34.8	97.0	4.8
	Over 1.80	3.0	78.4	100.0	7.0
48 to 100	Under 1.30	35.3	1.3	35.3	1.3
Weight, 13.0 pct	1.30 to 1.40	50.3	4.6	85.6	3.2
	1.40 to 1.60	9.4	15.9	95.0	4.5
	1.60 to 1.80	1.3	33.5	96.3	4.9
	Over 1.80	3.7	76.5	100.0	7.5
100 to 200	Under 1.30	23.7	1.4	23.7	1.4
Weight, 5.8 pct	1.30 to 1.40	57.2	4.4	80.9	3.5
	1.40 to 1.60	12.4	14.7	93.3	5.0
	1.60 to 1.80	1.5	31.8	94.8	5.4
	Over 1.80	5.2	73.7	100.0	9.0
Under 200	Under 1.30	10.1	4.2	10.1	4.2
Weight, 6.6 pct	1.30 to 1.40	58.5	5.0	68.6	4.9
	1.40 to 1.60	23.6	12.0	92.2	6.7
	1.60 to 1.80	1.9	26.1	94.1	7.1
	Over 1.80	5.9	70.1	100.0	10.8
Composite	Under 1.30	32.6	1.6	32.6	1.6
Weight, 100.0 pct	1.30 to 1.40	51.5	5.1	84.1	3.7
	1.40 to 1.60	10.6	16.2	94.7	5.1
	1.60 to 1.80	1.4	35.0	96.1	5.6
	Over 1.80	3.9	78.4	100.0	8.4

[a] Moisture-free basis.

Table III. Specific Gravity Analyses, by Size Fractions, of Clements, Ala., Coal

Size, Mesh	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
8 to 14	Under 1.30	87.5	1.4	87.5	1.4
Weight, 44.0 pct	1.30 to 1.40	6.4	6.6	93.9	1.8
	1.40 to 1.60	2.0	18.3	95.9	2.1
	1.60 to 1.80	0.8	42.2	96.7	2.4
	Over 1.80	3.3	74.6	100.0	4.8
14 to 28	Under 1.30	84.9	1.3	84.9	1.3
Weight, 28.3 pct	1.30 to 1.40	7.4	6.5	92.3	1.7
	1.40 to 1.60	2.2	17.5	94.5	2.1
	1.60 to 1.80	0.9	37.0	95.4	2.4
	Over 1.80	4.6	74.5	100.0	5.7
28 to 48	Under 1.30	77.5	1.3	77.5	1.3
Weight, 14.5 pct	1.30 to 1.40	10.7	5.8	88.2	1.8
	1.40 to 1.60	3.1	16.9	91.3	2.4
	1.60 to 1.80	1.3	34.2	92.6	2.8
	Over 1.80	7.4	76.9	100.0	8.3
48 to 100	Under 1.30	68.3	1.6	68.3	1.6
Weight, 7.8 pct	1.30 to 1.40	14.2	5.8	82.5	2.3
	1.40 to 1.60	4.0	16.5	86.5	3.0
	1.60 to 1.80	1.7	33.4	88.2	3.6
	Over 1.80	11.8	77.1	100.0	12.2
100 to 200	Under 1.30	57.0	2.1	57.0	2.1
Weight, 3.7 pct	1.30 to 1.40	20.7	5.7	77.7	3.1
	1.40 to 1.60	5.1	15.9	82.8	3.9
	1.60 to 1.80	2.1	31.9	84.9	4.5
	Over 1.80	15.1	74.1	100.0	15.0
Under 200	Under 1.30	29.4	5.9	29.4	5.9
Weight, 1.7 pct	1.30 to 1.40	36.4	6.5	65.8	6.2
	1.40 to 1.60	8.7	15.5	74.5	7.3
	1.60 to 1.80	3.2	30.9	77.7	8.3
	Over 1.80	22.3	71.9	100.0	22.5
Composite	Under 1.30	81.7	1.4	81.7	1.4
Weight, 100.0 pct	1.30 to 1.40	9.0	6.3	90.7	1.9
	1.40 to 1.60	2.6	17.3	93.3	2.3
	1.60 to 1.80	1.0	37.0	94.3	2.7
	Over 1.80	5.7	75.2	100.0	6.8

[a] Moisture-free basis.

is a moderate increase in the percentage of impurity with decrease in particle size, the material finer than 200-mesh containing about twice as much impurity as the material coarser than 14-mesh.

None of these coals contains enough material of intermediate specific gravity to be considered particularly difficult to wash.

Description of Spiral Installation

The laboratory installation of the spiral is shown in fig. 1. The unit itself is merely a spiral channel of modified semicircular cross-section. The type used for treating coal consists of six full turns in a vertical height of 5 ft with an outside diameter of 2 ft. A circular refuse port at the lowest point in the cross-section of the channel is provided at each 120° interval of spiral turn. A small launder paralleling the main channel carries wash water, which is introduced as a cross-flow at the inner edge of the stream below each refuse port.

In operation, coal and water at a solids concentration of 15 to 20 pct is introduced at the top of the spiral. As the coal flows downward, it is stratified in the stream according to particle size and specific gravity. The refuse product moves along the inner edge of the channel from which it is removed through the refuse ports. The proportion of refuse removed is controlled by the number of ports used and by adjustable splitters inserted in the ports. The remainder of the stream is discharged at the lower end of the spiral.

Where the spiral is used for treating ores, it is customary to draw a concentrate from the upper ports and a middling product from the lower ones. Optionally, a middling product can be cut from the inner portion of the stream discharge. Depending upon the character of the ore and the type of separation desired, either the tailing or the concentrate may be retreated in a secondary spiral; the middling product often is recirculated through the primary spiral. In treating coal any of these possibilities for retreatment could be utilized. However, in the two commercial plants now employing the spiral for cleaning coal, no retreatment is practiced—the spiral makes a two-product separation in a single pass of the material.

The laboratory spiral, as indicated in fig. 1, is a full-size unit installed in closed circuit with a centrifugal pump and a pump sump to permit rapid testing of small samples. Part of the output of the pump is sent through a centrifuge to provide clarified wash water for this closed-circuit arrangement.

Test Procedure

In making a test, the pump sump was filled to a fixed level with water, an 8-kg sample of coal was added to the sump, and the pulp was allowed to circulate through the spiral for 2 or 3 min to insure achieving equilibrium conditions. Simultaneous time samples were then collected from the individual refuse ports and from the end discharge of the spiral. In most of the work a splitter was employed to divide the end discharge stream into a washed coal and a middling product, the middling product comprising the inner portion of the stream.

Of the 18 refuse ports provided, only five—one for each of the first five turns of the spiral—were employed. In a typical series of tests on a particular coal, three or four trials were made with different settings of the splitters in the refuse ports in order to determine the optimum conditions required to give a refuse product of the desired quality. Then, with the refuse port openings fixed, several tests

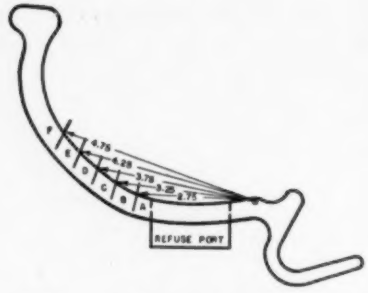

Fig. 2—Cross section of spiral showing zones in stream created by various positions of washed coal-middling splitter.

were made with different settings of the splitter that divided the end-discharge stream into washed coal and middling in order to establish the proper position for this division. With proper operating conditions thus established, a final test was made to obtain samples of all products for float-and-sink and screen-sizing examination.

A middling product was made in nearly all of the tests to obtain more complete information on spiral performance. It was realized that a three-product separation is not desirable in most coal-washing operations. However, it was thought that in this work a middling product should be produced and its character in terms of size and specific-gravity composition determined in order to show the possibilities inherent in a three-product separation. Little work was done towards retreating this middling material, and at the present it is not possible to state definitely whether the middling material should be

Table IV. Specific Gravity Analyses, by Size Fractions, of Poplar Ridge, Ky., Coal

Size, Mesh	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
8 to 14	Under 1.30	42.2	3.9	42.2	3.9
Weight, 21.6 pct	1.30 to 1.40	41.2	10.3	83.4	7.1
	1.40 to 1.60	9.8	21.1	93.2	8.5
	1.60 to 1.80	1.1	36.2	94.3	8.9
	Over 1.80	5.7	65.4	100.0	12.1
14 to 28	Under 1.30	35.1	3.2	35.1	3.2
Weight, 32.3 pct	1.30 to 1.40	39.9	9.5	75.0	6.6
	1.40 to 1.60	11.8	19.7	86.8	8.3
	1.60 to 1.80	1.9	36.7	88.7	8.9
	Over 1.80	11.3	69.3	100.0	15.8
28 to 48	Under 1.30	33.3	3.0	33.3	3.0
Weight, 19.3 pct	1.30 to 1.40	36.8	9.0	70.1	6.1
	1.40 to 1.60	13.2	19.2	83.3	8.2
	1.60 to 1.80	2.6	37.3	85.9	9.1
	Over 1.80	14.1	69.7	100.0	17.6
48 to 100	Under 1.30	25.0	3.1	25.0	3.1
Weight, 12.0 pct	1.30 to 1.40	38.5	8.2	63.5	6.2
	1.40 to 1.60	16.5	18.7	80.0	8.8
	1.60 to 1.80	4.0	37.1	84.0	10.1
	Over 1.80	16.0	68.3	100.0	19.4
100 to 200	Under 1.30	14.7	3.5	14.7	3.5
Weight, 5.3 pct	1.30 to 1.40	38.7	7.6	53.4	6.5
	1.40 to 1.60	21.3	17.5	74.7	9.6
	1.60 to 1.80	6.1	35.1	80.8	11.5
	Over 1.80	19.2	66.9	100.0	22.2
Under 200	Under 1.30	3.9	7.1	3.9	7.1
Weight, 9.5 pct	1.30 to 1.40	23.2	7.5	27.1	7.4
	1.40 to 1.60	31.0	14.6	58.1	11.3
	1.60 to 1.80	16.7	27.7	74.8	14.9
	Over 1.80	25.2	63.2	100.0	27.1
Composite	Under 1.30	31.0	3.4	31.0	3.4
Weight, 100.0 pct	1.30 to 1.40	37.7	9.2	68.7	6.6
	1.40 to 1.60	14.6	18.5	83.3	8.7
	1.60 to 1.80	3.8	33.1	87.1	9.7
	Over 1.80	12.9	67.6	100.0	17.2

[a] Moisture-free basis.

recirculated through the spiral or whether it should be retreated by a hydraulic classifier or other suitable means to recover its content of clean coal.

All spiral tests were made at a feed rate of 1 ton of coal per hr and at a concentration of solids in the feed of 18 to 20 pct. The influence of these two variables—solids concentration and feed rate—were not investigated thoroughly because previous published work on the spiral[8, 9] appeared to have demonstrated conclusively that the optimum concentration of solids in the feed was 15 to 20 pct and that feed rates in excess of about 1 ton per hr resulted in impaired efficiency.

Black Eagle, W. Va., Coal

The results of a series of tests made to determine the influence of refuse-port settings with Black Eagle coal are presented in table VI.

All were made with the use of one refuse port in each of the top five turns of the spiral; the ports are numbered from the top down. With these five ports three fourths open, as in test 26, the refuse amounted to 11 pct of the feed and contained 23.4 pct ash. Closing the top two ports to one half open and the bottom three ports to one fourth open, as in test 39, reduced the amount of refuse to 4.6 pct of the feed and increased the grade to 39.4 pct ash. Closing the ports still further, as was done in test 37, reduced the production of refuse to only 2.3 pct but increased the grade of refuse to only 43.3 pct ash. Thus, the port openings prevailing in test 39 were considered optimum, even though the ash content of the refuse product was less than 40 pct. The 4.6 pct yield of refuse in this test is a little less than the 5.3 pct of impurity heavier than 1.60 sp gr present in the raw coal. For comparison, the impurity heavier than 1.60 sp gr has an ash content of 66.9 pct.

Table V. Specific Gravity Analyses, by Size Fractions, of Roslyn, Wash., Coal

Size, Mesh	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
8 to 14	Under 1.30	39.6	5.9	39.6	5.9
Weight, 31.3 pct	1.30 to 1.40	37.0	11.3	76.6	8.5
	1.40 to 1.60	7.6	26.6	84.2	10.1
	1.60 to 1.80	2.7	43.7	86.9	11.2
	Over 1.80	13.1	79.0	100.0	20.1
14 to 28	Under 1.30	37.2	5.5	37.2	5.5
Weight, 31.4 pct	1.30 to 1.40	36.9	10.7	74.1	8.1
	1.40 to 1.60	9.3	24.9	83.4	10.1
	1.60 to 1.80	3.2	42.9	86.6	11.2
	Over 1.80	13.4	78.9	100.0	20.3
28 to 48	Under 1.30	28.9	5.2	28.9	5.2
Weight, 13.2 pct	1.30 to 1.40	41.5	10.5	70.4	8.3
	1.40 to 1.60	10.2	25.0	80.6	10.4
	1.60 to 1.80	3.0	45.9	83.6	11.7
	Over 1.80	16.4	74.5	100.0	22.0
48 to 100	Under 1.30	23.2	4.8	23.2	4.8
Weight, 11.8 pct	1.30 to 1.40	40.9	10.3	64.1	8.3
	1.40 to 1.60	13.3	23.9	77.4	11.0
	1.60 to 1.80	4.2	43.3	81.6	12.7
	Over 1.80	18.4	69.9	100.0	23.2
100 to 200	Under 1.30	18.4	4.5	18.4	4.5
Weight, 7.7 pct	1.30 to 1.40	40.9	10.1	59.3	8.4
	1.40 to 1.60	15.5	22.4	74.8	11.3
	1.60 to 1.80	4.8	41.9	79.6	13.1
	Over 1.80	20.4	67.0	100.0	24.1
Under 200	Under 1.30	9.5	5.2	9.5	5.2
Weight, 4.6 pct	1.30 to 1.40	35.1	9.7	44.6	8.7
	1.40 to 1.60	22.9	18.7	67.5	12.1
	1.60 to 1.80	5.5	36.7	73.0	14.0
	Over 1.80	27.0	65.1	100.0	27.8
Composite	Under 1.30	32.5	5.5	32.5	5.5
Weight, 100.0 pct	1.30 to 1.40	38.2	10.7	70.7	8.3
	1.40 to 1.60	10.5	24.2	81.2	10.4
	1.60 to 1.80	3.4	42.9	84.6	11.7
	Over 1.80	15.4	74.7	100.0	21.4

[a] Moisture-free basis.

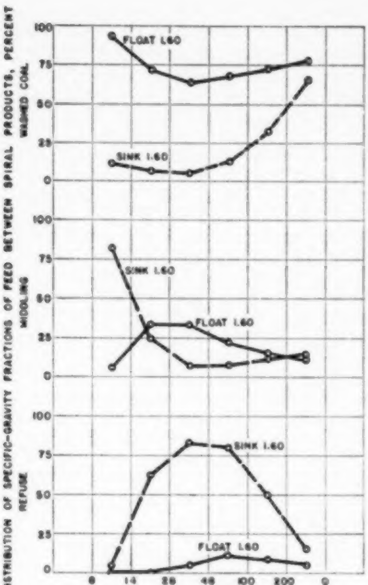

Fig. 3—Percentage distribution of the specific gravity fractions of Black Eagle coal between spiral products by size fractions.

The results presented in table VI demonstrate that with this coal the spiral was unable to effect a separation between coal and impurity accurately enough to give a refuse product of the quality generally considered acceptable.

Table VII shows the results of a series of five tests made with this coal to determine the influence of the split between washed coal and middling. For this series of tests, the washed coal-middling splitter was set according to the diagram shown in fig. 2. With the splitter set at 2.75 in., a washed coal amounting to 83 pct of the feed and containing 5.8 pct of ash was produced. With progressively increasing amounts of material directed into the middling product, as in tests 43, 44, and 45, the ash content of the washed coal was reduced steadily to a minimum of 4.9 pct. In test 46, however, in which the splitter was set to include in the middling all but the extreme outer portion of the stream, the ash content of the washed coal increased to 5.8 pct. As the increase in the proportion of middling was considered excessive for the accompanying reduction in the ash content of the washed coal, test 42 was considered to represent the optimum division between washed coal and middling.

Table VIII shows the results of float-and-sink tests made at 1.60 sp gr on the spiral products of test 42, the test in which the yield of washed coal was 83.0 pct and yield of middling 13.3 pct.

The washed coal contained 1.1 pct of impurity heavier than 1.60 sp gr, and the presence of this impurity increased its ash content by 0.5 pct. The refuse product contained over 50 pct of coal lighter than 1.60 sp gr. The middling product contained roughly three times the amount of heavy impurity present in the feed.

Screen analyses of these float-and-sink fractions

Table VI. Influence of Port Openings on Amount and Quality of Refuse, Black Eagle Coal

Product	Port Opening	Weight, Pct	Ash,[a] Pct	Cumulative Weight Pct	Cumulative Ash,[a] Pct
Test 26					
Refuse port 1	3/4	1.6	39.1	1.6	39.1
2	3/4	3.6	32.8	5.2	34.7
3	3/4	1.6	15.7	6.8	30.3
4	3/4	1.9	12.4	8.7	26.4
5	3/4	2.3	12.3	11.0	23.4
End discharge		89.0	6.7	100.0	8.5
Test 39					
Refuse port 1	1/2	1.1	42.4	1.1	42.4
2	1/2	2.0	48.3	3.1	46.2
3	1/4	0.4	37.8	3.5	45.2
4	1/4	0.4	22.1	3.9	42.9
5	1/4	0.7	20.0	4.6	39.4
End discharge		95.4	7.4	100.0	8.9
Test 37					
Refuse port 1	3/8	0.5	32.1	0.5	32.1
2	3/8	1.2	54.9	1.7	48.2
3	1/8	0.3	33.4	2.0	46.0
4	1/8	0.1	32.8	2.1	45.3
5	1/8	0.2	22.0	2.3	43.3
End discharge		97.7	7.2	100.0	8.0

[a] Moisture-free basis.

Table VII. Influence of Split Between Washed Coal and Middling, Black Eagle Coal

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
42	Washed coal	2.75	83.0	5.8	83.0	5.8
	Middling		13.3	16.0	96.3	7.2
	Refuse		3.7	40.3	100.0	8.4
43	Washed coal	3.25	67.5	5.2	67.5	5.2
	Middling		28.7	11.2	96.2	7.0
	Refuse		3.8	40.3	100.0	8.3
44	Washed coal	3.75	45.7	5.0	45.7	5.0
	Middling		50.3	10.1	96.0	7.7
	Refuse		4.0	40.3	100.0	9.0
45	Washed coal	4.25	23.7	4.9	23.7	4.9
	Middling		72.4	8.3	96.1	7.5
	Refuse		3.9	40.3	100.0	8.7
46	Washed coal	4.75	7.4	5.8	7.4	5.8
	Middling		88.6	7.1	96.0	7.0
	Refuse		4.0	40.3	100.0	8.3

[a] Moisture-free basis.

Table VIII. Float-and-Sink Separations at 1.60 Sp Gr on Products from Spiral Test of Black Eagle Coal

Product	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
Washed coal	Under 1.60	98.9	5.2	98.9	5.2
Weight, 83.0 pct	Over 1.60	1.1	53.4	100.0	5.7
Middling	Under 1.60	84.6	6.3	84.6	6.3
Weight, 13.3 pct	Over 1.60	15.4	69.3	100.0	16.0
Refuse	Under 1.60	52.6	7.7	52.6	7.7
Weight, 3.7 pct	Over 1.60	47.4	78.9	100.0	41.4

[a] Moisture-free basis.

Table IX. Screen Analyses of Specific Gravity Fractions of Spiral Products of Black Eagle Coal, Pct

Product	Weight, Pct	Screen Size, Mesh 8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	83.0						
Under 1.60	98.9	48.3	26.3	11.5	6.3	3.5	4.1
Over 1.60	1.1	55.8	9.7	4.4	5.3	8.8	16.0
Middling	13.3						
Under 1.60	84.6	6.5	42.5	31.1	12.6	3.9	3.4
Over 1.60	15.4	69.0	21.6	3.3	2.0	1.6	2.5
Refuse	3.7						
Under 1.60	52.6	0.0	4.2	32.5	35.3	14.8	13.2
Over 1.60	47.4	4.5	41.7	31.7	16.3	4.0	1.8

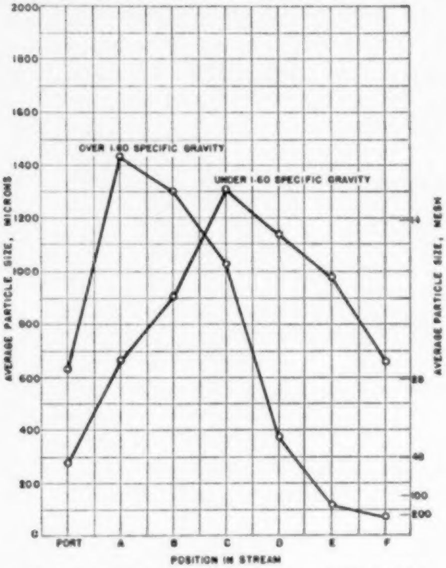

Fig. 4—Average particle size of specific gravity fractions of feed carried in various portions of stream.

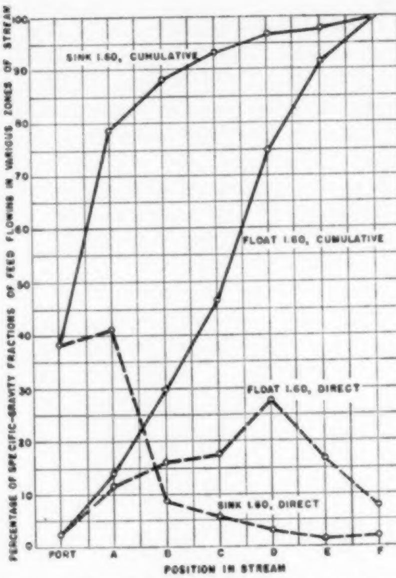

Fig. 5—Percentage of specific gravity fractions of feed carried in various portions of stream.

reveal some interesting aspects of spiral performance, as indicated in table IX. The impurity in the washed coal was principally coarse material, 55.8 pct being in the 8 to 14-mesh size fraction. The coal contaminating the refuse product, on the other hand, contained no material coarser than 14-mesh and bulked largely between 28 and 100-mesh. The coal in the middling product was noticeably finer than that in the washed coal, concentrating largely in the range from 14 to 48-mesh. The 15 pct of heavy impurity associated with this coal in the middling product was distinctly coarse material; nearly 70 pct was in the 8 to 14-mesh size fraction.

These figures thus demonstrate that the spiral, like most other gravity concentrating devices, effects a separation by particle size as well as by specific gravity.

Table X. Screen Analyses of Specific Gravity Fractions Flowing in Various Zones of Spiral Stream, Pct.

Product and Specific Gravity	Weight, Pct	Screen Size, Mesh 8-14	14-28	28-48	48-100	100-200	Under 200
Refuse	3.9						
Under 1.60	52.6	0.0	4.2	32.5	35.3	14.8	13.2
Over 1.60	47.4	4.5	41.7	31.7	16.3	4.0	1.8
Zone A	13.2						
Under 1.60	84.6	6.6	42.4	30.9	12.8	3.9	3.4
Over 1.60	15.4	68.7	21.7	3.4	2.1	1.6	2.5
Zone B	15.5						
Under 1.60	97.2	23.3	42.4	21.4	7.2	3.1	2.6
Over 1.60	2.8	62.2	17.2	5.7	4.6	4.6	5.7
Zone C	16.9						
Under 1.60	98.3	55.2	32.0	8.7	3.3	0.7	0.1
Over 1.60	1.7	48.3	11.7	6.7	8.3	10.0	15.0
Zone D	26.7						
Under 1.60	99.4	46.1	28.4	11.0	6.2	1.8	6.5
Over 1.00	0.6	9.4	12.5	9.4	12.5	15.6	40.6
Zone E	16.3						
Under 1.60	99.7	38.4	20.5	16.3	12.1	7.9	4.8
Over 1.60	0.3	0.0	6.1	6.1	17.5	26.3	43.9
Zone F	7.5						
Under 1.60	98.9	26.2	7.4	11.4	16.8	14.8	23.5
Over 1.60	1.1	0.0	1.2	1.2	6.1	20.2	71.2

The influence of particle size in determining the recovery of the specific gravity fractions of the feed in the various spiral products is illustrated clearly in fig. 3. These curves indicate, for example, that of the total 8 to 14-mesh coal lighter than 1.60 sp gr present in the feed, 94 pct was recovered in the washed coal. The corresponding recovery decreased to reach a minimum of 62 pct in the 28 to 48-mesh size, then increased again with further decrease in particle size. Similarly, of the 8 to 14-mesh impurity heavier than 1.60 sp gr present in the spiral feed, only 4 pct went into the refuse product, 84 pct was discharged as middling, and 12 pct entered the washed coal. The elimination of heavy impurity in the refuse product reached a maximum of 85 pct in the 28 to 48-mesh size and then decreased again with further decrease in particle size.

The influence of particle size on the performance of the spiral is illustrated even more clearly in the results of a series of tests on Black Eagle coal in which the character of the material flowing in each ½-in. increment of the stream was determined. Operating conditions for this series of tests were the same as those prevailing in test 42, which was described previously. The zones or increments of the spiral stream used in this work are illustrated in fig. 2. Zone A included all of the innermost portion of the stream except the material drawn off through the refuse ports. Zones B through E were approximately ½ in. wide, as measured along the cross-section of the spiral, and zone F included all of the stream beyond E.

Table X shows the proportion of the spiral feed flowing in each zone of the stream, the percentages of materials heavier and lighter than 1.60 sp gr in each zone, and the screen analyses of these specific

gravity fractions. As shown by the tabular data in table X and by the graphic presentation in fig. 4, the clean coal lighter than 1.60 sp gr becomes progressively coarser with greater distance from the inside of the spiral until maximum size is achieved in zone C. The clean coal flowing in the stream beyond this point becomes progressively finer.

The distribution of heavy impurity in the stream differs markedly from that of the clean coal. As shown by the data in table X and also by the graphic presentation in fig. 5, the material flowing in zone A just outside the refuse ports contains a high proportion of impurity. In fact, reference to fig. 4 indicates that, whereas 38 pct of the total impurity in the feed was stratified along the inner edge of the stream where it could be drawn through the refuse ports, 41 pct of the impurity was stratified just a little farther out in the stream where it could not be caught in the ports. The coarseness of the impurity flowing in zone A accounts for its presence that far out in the stream; the impurity in this zone contained 68.7 pct of material coarser than 14-mesh, while the impurity removed through the refuse ports contained only 4.5 pct in this size fraction. The proportion of the total impurity accounted for in the other zones of the stream decreases rather rapidly to reach a minimum of 1.0 pct in zone E. The particle size of the impurity also decreases rapidly in these zones.

To generalize on the performance of the spiral, the modifying influence of size on specific gravity is such that intermediate-size particles tend to stratify in the inner portion of the stream, coarser particles are stratified farther out, and the finest particles remain suspended in the water to assemble predominantly in the extreme outer portion of the stream. In consequence of this behavior, the washed coal is contaminated with impurity principally in the coarsest and finest sizes, while conversely, the major loss of clean coal in the refuse product occurs in the intermediate sizes.

The magnitude of this loss of coal to the refuse product in the intermediate sizes, in terms of the efficiency of a two-product separation of this coal, is indicated in table XI. Efficiency, as the term is used here, is the ratio of the yield of washed coal to the yield of float coal of the same ash content shown to be present in the feed by specific gravity analysis. As indicated in table XI, operation of the spiral to reduce the ash content of this coal from 8.4 to 7.2 pct by the removal of 3.7 pct of refuse resulted in an efficiency of 97.4 pct. If greater ash reduction is attempted in a two-product separation, efficiency decreases rapidly. Producing a washed product of 5.2 pct ash, corresponding to a separation at 1.60 sp gr, results in an efficiency of only 71 pct.

For the sake of comparison, this coal unquestionably could be washed at 1.60 sp gr on a table with an efficiency of 99 pct.

Clements, Ala., Coal

The Clements coal contains slightly more heavy impurity than that in the Black Eagle coal already described, and the impurity is considerably finer. From the standpoint of specific gravity composition, the Clements coal is very easy to wash since it contains little material of intermediate density. Inasmuch as the behavior of these two coals in the spiral was quite similar, less detailed data will be presented for the Clements coal.

Table XII indicates the results obtained in two trials made to determine the optimum settings of the refuse ports. In test 40, made with the upper two ports one half open and the three lower ports one fourth open, a refuse product amounting to 3.8 pct and containing 49.6 pct ash was produced. Removing a greater percentage of refuse would have been desirable for, as shown by the float-and-sink data in table III, the raw coal contained 6.7 pct impurity heavier than 1.60 sp gr. However, when the ports were opened wider for test 41, the quality of the refuse was decreased from nearly 50 pct ash to 36 pct. In the light of these results, the port openings shown for test 40 were adopted as standard for all subsequent work.

The results obtained with various settings of the washed coal-middling splitter are presented in table XIII. A washed coal containing 3.4 pct ash was obtained at a yield of 86.4 pct in test 47, in which a middling product amounting to 10.1 pct of the feed was made. As with Black Eagle, moving the splitter farther out into the stream gave a washed product of somewhat lower ash content but only with drasti-

Table XI. Efficiency of Washing Black Eagle Coal to Various Ash Contents

Test No.	Ash[a] in Washed Coal, Pct	Yield of Washed Coal, Pct	Yield of Float Coal, Pct	Efficiency, Pct
42	7.2	96.3	98.9	97.4
26	6.7	89.0	98.3	90.5
42	5.8	83.0	96.7	85.8
43	5.2	67.5	95.0	71.1

[a] Moisture-free basis.

Table XII. Influence of Port Openings on Amount and Quality of Refuse, Clements Coal

Product	Port Opening	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
Test 40					
Refuse port 1	½	1.2	50.4	1.2	50.4
2	½	1.5	59.5	2.7	55.5
3	¼	0.3	52.8	3.0	55.2
4	¼	0.3	30.6	3.3	53.0
5	¼	0.5	27.5	3.8	49.6
End discharge		96.2	4.2	100.0	5.9
Test 41					
Refuse port 1	¾	2.3	43.4	2.3	43.4
2	¾	2.4	43.5	4.7	43.5
3	½	0.6	32.4	5.3	42.2
4	½	0.7	20.7	6.0	39.7
5	½	1.1	15.5	7.1	35.9
End discharge		92.9	3.9	100.0	6.1

[a] Moisture-free basis.

Table XIII. Influence of Split Between Washed Coal and Middling, Clements Coal

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
47	Washed coal	2.75	86.4	3.4	86.4	3.4
	Middling		10.1	12.4	96.5	4.3
	Refuse		3.5	49.7	100.0	5.9
48	Washed coal	3.25	75.5	2.9	75.5	2.9
	Middling		20.9	10.8	96.4	4.6
	Refuse		3.6	49.5	100.0	6.2
49	Washed coal	3.75	56.9	3.1	56.9	3.1
	Middling		39.7	6.7	96.6	4.6
	Refuse		3.4	49.5	100.0	6.1
50	Washed coal	4.25	32.4	3.3	32.4	3.3
	Middling		64.2	5.5	96.6	4.8
	Refuse		3.4	49.5	100.0	6.3
51	Washed coal	4.75	21.5	2.6	21.5	2.6
	Middling		75.1	5.6	96.6	4.9
	Refuse		3.4	49.5	100.0	6.4

[a] Moisture-free basis.

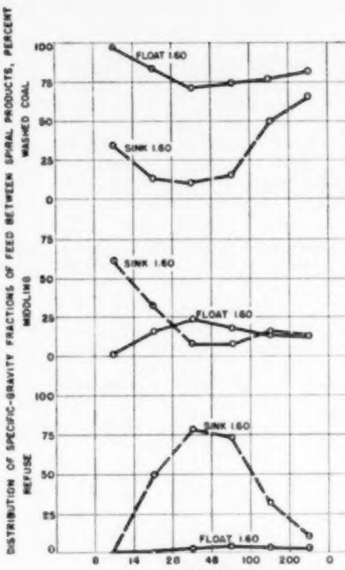

Fig. 6—Percentage distribution of the specific gravity fractions of Clements coal between spiral products by size fractions.

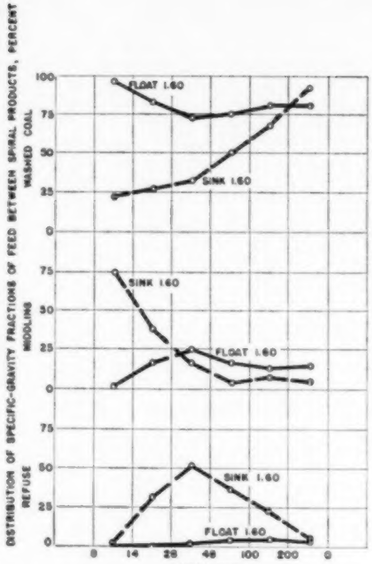

Fig. 7—Percentage distribution of the specific gravity fractions of Poplar Ridge coal between spiral products by size fractions.

cally reduced yield. For example, comparing tests 47 and 48, to reduce the ash content of the washed coal from 3.4 to 2.9 pct, the percentage of the feed directed into the middling product had to be increased from 10 to 21 pct. Accordingly, the conditions prevailing in test 47 were considered optimum, and samples of all products made in this test were subjected to float-and-sink and screen-sizing examination.

The results of this examination are shown in table XIV.

The washed coal contained 1.9 pct of impurity heavier than 1.60 sp gr, and the presence of this impurity increased its ash content from 2.3 to 3.4 pct. The refuse contained 42.1 pct of coal lighter than 1.60 sp gr, and the middling product was composed of 85 pct coal and 15 pct impurity. Screen analyses of these specific gravity fractions are given in table XV.

The impurity contaminating the washed coal was substantially finer than that in the washed product from Black Eagle, not because of any difference in spiral operation but rather because of the fineness of the impurity in the raw coal. As with Black Eagle,

Table XIV. Float-and-Sink Separations at 1.60 Sp Gr on Products from Spiral Test on Clements Coal

Product	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
Washed coal	Under 1.60	98.1	2.3	98.1	2.3
Weight, 86.4 pct	Over 1.60	1.9	58.3	100.0	3.4
Middling	Under 1.60	85.5	3.0	85.5	3.0
Weight, 10.1 pct	Over 1.60	14.5	68.1	100.0	12.4
Refuse	Under 1.60	42.1	4.8	42.1	4.8
Weight, 3.5 pct	Over 1.60	57.9	82.4	100.0	49.7

[a] Moisture-free basis.

the heavy impurity in the middling product was substantially coarser than that in the washed coal, and the impurity in both was much coarser than that drawn off through the refuse ports. The clean coal lost in the refuse product tended to be of intermediate size.

Fig. 6 illustrates the magnitude of the influence of particle size by showing the recovery in each spiral product of the specific gravity and size fractions present in the feed to the spiral. The curves in fig. 6 do not differ materially from those given in fig. 3 for Black Eagle coal. They show the same maximum elimination of impurity in the intermediate sizes and the same maximum loss of clean coal in the intermediate sizes.

On the whole, operation of the spiral on Clements coal was less satisfactory than on Black Eagle. The specific gravity analyses of Clements coal given in table III indicate that a reduction in ash content from 6.8 to 2.3 pct should be obtainable with high efficiency. The spiral was unable to produce a washed product containing 2.3 pct ash, and, even in making a washed product of substantially higher ash content, an appreciable loss of clean coal occurred.

Poplar Ridge, Ky., Coal

The Poplar Ridge coal is much more difficult to wash than either of the two coals described previously because it contains substantially more material of intermediate density and a great deal more heavy impurity. This coal might be described as moderately difficult to wash, while on the same basis the two previous coals would be described as very easy to wash. Considering all sizes together, this coal contains 16.7 pct of impurity heavier than 1.60 sp gr, and the proportion of impurity increases steadily with decrease in particle size. The sizes finer than 100-mesh contain over 30 pct of impurity.

The results of two trials made to determine the

Table XV. Screen Analyses of Specific Gravity Fractions of Spiral Products of Clements Coal, Pct

Product	Weight, Pct	Screen Size, Mesh 8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	86.4						
Under 1.60	98.1	57.5	23.5	9.6	5.4	2.4	1.6
Over 1.60	1.9	28.9	8.6	5.9	7.0	12.8	36.8
Middling	10.1						
Under 1.60	85.5	7.0	39.9	32.3	14.3	4.3	2.2
Over 1.60	14.5	53.9	25.7	5.2	4.5	5.0	5.7
Refuse	3.5						
Under 1.60	42.2	9.1	10.7	33.6	29.2	11.0	6.4
Over 1.60	57.8	1.3	28.1	35.9	24.3	6.5	3.9

Table XVI. Influence of Port Openings on Amount and Quality of Refuse, Poplar Ridge Coal

Product	Port Opening	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
Test 24					
Refuse port 1	3/4	3.0	47.6	3.0	47.6
2	3/4	4.2	45.2	7.2	46.2
3	1/2	1.0	40.5	8.2	45.5
4	1/2	1.2	32.8	9.4	43.9
5	1/2	1.6	25.0	11.0	41.1
End discharge		89.0	11.5	100.0	14.8
Test 25					
Refuse port 1	1/2	1.5	52.3	1.5	52.3
2	1/2	3.5	51.9	5.0	52.0
3	1/2	0.7	44.0	5.7	51.0
4	1/4	0.6	43.8	6.3	50.3
5	1/4	0.5	35.0	6.8	49.2
End discharge		93.1	11.9	100.0	14.5

[a] Moisture-free basis.

optimum opening of the refuse ports are shown in table XVI. In test 25, made with the top three refuse ports one half open and the lower two ports one fourth open, 6.8 pct of refuse containing about 50 pct ash was produced. This quantity of refuse is less than one half of the amount of impurity heavier than 1.60 sp gr shown to be present in the feed by the specific gravity analysis given in table IV. However, as demonstrated by the results of test 24, opening the refuse ports wider to extract a higher percentage of refuse impaired greatly the quality of this product. The port openings of test 25 were therefore adopted as optimum for subsequent work.

The results of five tests made to explore the influence of the split between washed coal and middling are given in table XVII. It will be observed that with this coal the production of a middling product was not an effective means of reducing the ash content of the washed coal. In test 33, for example, directing 14 pct of the feed into a middling product reduced the ash content of the washed coal by only 1.1 pct. Increasing the cut to middling, as was done in the other four tests of this series, had substantially no influence on the quality of the resulting washed coal. Consequently, the remainder of the performance data for this coal are presented in terms of a two-product separation.

Table XVIII shows the results of float-and-sink separations made at 1.60 sp gr on washed coal and refuse produced by the spiral when operated with the refuse port openings indicated for test 25. The washed coal contained nearly 15 pct of impurity heavier than 1.60 sp gr, and the presence of this impurity increased the ash content of the washed coal from 7.9 to 12.6 pct. The refuse had an ash content of about 54 pct and contained 21.2 pct of coal lighter than 1.60 sp gr.

Screen analyses of the specific gravity fractions of table XVIII are given in table XIX. The heavy impurity in the washed coal was predominantly fine material, 54 pct being finer than 200-mesh. The clean coal lost in the refuse product was largely of 28 to 100-mesh size, and, in this respect, performance of the spiral on this coal was the same as on the other two coals previously described.

Table XVII. Influence of Split Between Washed Coal and Middling, Poplar Ridge Coal

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
33	Washed coal	2.75	79.4	11.8	79.4	11.8
	Middling		14.0	19.1	93.4	12.9
	Refuse		6.6	53.7	100.0	15.6
29	Washed coal	3.25	65.2	12.0	65.2	12.0
	Middling		28.7	14.0	93.9	12.6
	Refuse		6.1	51.6	100.0	15.0
30	Washed coal	3.75	44.7	11.9	44.7	11.9
	Middling		49.4	12.8	94.1	12.4
	Refuse		5.9	53.7	100.0	14.8
31	Washed coal	4.25	23.2	16.1	23.2	16.1
	Middling		70.9	11.9	94.1	12.9
	Refuse		5.9	52.6	100.0	15.3
32	Washed coal	4.75	10.8	16.4	10.8	16.4
	Middling		83.2	12.6	94.0	13.0
	Refuse		6.0	54.3	100.0	15.5

[a] Moisture-free basis.

Table XVIII. Float-and-Sink Separations at 1.60 Sp Gr on Products from Spiral Test of Poplar Ridge Coal

Product	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
Washed coal	Under 1.60	85.1	7.9	85.1	7.9
Weight, 94.3 pct	Over 1.60	14.9	39.6	100.0	12.6
Refuse	Under 1.60	21.2	13.0	21.2	13.0
Weight, 5.7 pct	Over 1.60	78.8	64.9	100.0	53.9

[a] Moisture-free basis.

Table XIX. Screen Analyses of Specific Gravity Fractions of Spiral Products of Poplar Ridge Coal, Pct

Product	Weight, Pct	Screen Size, Mesh 8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	94.3						
Under 1.60	85.1	28.9	38.0	17.4	9.4	4.0	2.3
Over 1.60	14.9	6.4	17.9	9.6	8.1	7.6	50.4
Refuse	5.7						
Under 1.60	21.2	0.0	4.1	37.9	35.2	15.2	7.6
Over 1.60	78.8	0.7	30.8	38.1	17.7	6.3	6.4

Fig. 7, which shows the distribution made by the spiral of each size and specific gravity fraction of the feed between the various spiral products, differs from the corresponding curves given for Black Eagle and Clements coals in only one particular. With this coal the proportion of heavy impurity entering the washed product increased steadily with decrease in particle size instead of reaching a minimum in the intermediate sizes. This circumstance probably is attributable to the fineness of the impurity in the Poplar Ridge coal.

With a coal containing so much fine impurity, desliming of the washed coal from the spiral would have reduced its ash content substantially.

Roslyn, Wash., Coal

Roslyn coal, like Poplar Ridge, contains considerable heavy impurity and a moderate amount of material of intermediate density. Although the finer

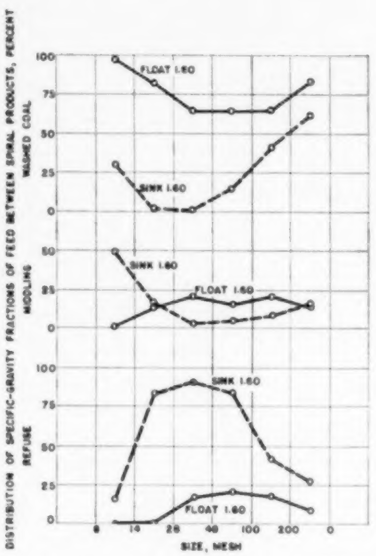

Fig. 8—Percentage distribution of the specific gravity fractions of Roslyn coal between spiral products by size fractions.

sizes are somewhat dirtier than the coarse material, this trend is not so pronounced as in the Poplar Ridge coal.

A great many spiral tests have been made with Roslyn coal, and no difficulty was experienced in setting the refuse ports to obtain a refuse product ranging from 56 to 59 pct ash. Refuse of better grade than this could not be made in appreciable quantity. Incidentally, with this coal, it was found that the same quantity and grade of refuse could be produced using only the top three ports as by using five ports, all only partly open. Thus, in making a two-product separation on Roslyn coal, the spiral could be shortened from six turns to three turns without any sacrifice in efficiency.

The results of a series of five trials made to determine the influence of the washed coal-middling splitter setting are given in table XX. The ash content of the washed coal was reduced from 14.3 to 11.8 pct by making a middling product amounting to 12 pct of the feed. However, doubling the amount of material directed into the middling product only reduced the ash in the washed coal an additional 0.6 pct.

Table XX. Influence of Split Between Washed Coal and Middling, Roslyn Coal

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
60	Washed coal	2.75	71.8	11.8	71.8	11.8
	Middling		12.2	29.0	84.0	14.3
	Refuse		16.0	58.8	100.0	21.4
61	Washed coal	3.25	60.3	11.2	60.3	11.2
	Middling		24.4	23.4	84.7	14.7
	Refuse		15.3	57.5	100.0	21.3
62	Washed coal	3.75	44.0	11.2	44.0	11.2
	Middling		39.3	17.8	83.3	14.3
	Refuse		16.7	59.3	100.0	21.8
63	Washed coal	4.25	22.0	12.9	22.0	12.9
	Middling		61.7	15.7	83.7	15.0
	Refuse		16.3	57.0	100.0	21.8
64	Washed coal	4.75	10.5	16.8	10.5	16.8
	Middling		74.0	15.1	84.5	15.3
	Refuse		15.5	56.5	100.0	21.7

[a] Moisture-free basis.

To obtain samples for float-and-sink tests, a trial was made with the spiral adjustments prevailing in test 60 of table XX. The spiral products from this test were deslimed at 200-mesh to eliminate a part of the clay slimes that form through the disintegration of the clay associated with Roslyn coal. The slimes removed amounted to 3.9 pct of the spiral feed and contained 50 pct ash. The deslimed spiral products were separated at 1.60 sp gr. The results are given in table XXI. The washed coal contained 3.7 pct of impurity heavier than 1.60 sp gr, and the presence of this material increased its ash content from 9.6 to 11.4 pct. The refuse contained about 28 pct of coal lighter than 1.60 sp gr, and the middling product was composed of 70 pct coal accompanied by 30 pct impurity.

Screen analyses of these float-and-sink fractions are given in table XXII. As with the other coals tested, the impurity contaminating the washed coal was principally material coarser than 14-mesh, together with slimes finer than 100-mesh. Conversely, the coal in the refuse product bulked largely between 28 and 100-mesh. The coal in the middling was substantially finer than the accompanying impurity.

The performance of the spiral on Roslyn coal is summarized best in fig. 8, which shows how the specific gravity fractions of the spiral feed were divided into washed coal, middling, and refuse in each size fraction. Just as with the other coals described previously, the recovery of clean coal was best in the coarsest and finest sizes, but the elimination of impurity was poorest in these sizes.

Table XXIII was prepared to show the type of two-product separation possible with the spiral on this coal. These data are from the test described previously in which the spiral products were deslimed; the two-product separation was created by adding the middling to the washed coal. In the 8 to 14-mesh size, little ash reduction was effected because this size of material reports almost wholly to the washed product. The spiral performed best on the 14 to 28-mesh size in which the ash was reduced from 20.4 to 11.1 pct with an efficiency of 97.6 pct. In the 28 to 48 and 48 to 100-mesh sizes the efficiency was lowest, for these are the sizes of coal that characteristically stratify with the refuse product and are thus lost. In the sizes under 100-mesh, efficiency improved, but ash reduction suffered because most of the fine material tended to accompany the washed coal.

Spiral Test on Classified Roslyn Coal

The performance data included in this report for four coals having materially different washability characteristics suggest that the ideal feed for the spiral—to permit it to make maximum ash reduction with minimum loss of coal—would be one in which the clean coal was of 8 to 28-mesh size and the accompanying impurity was of 28 to 100-mesh size. These ideal specifications could not be met by any natural raw coal, but they are approached by a hydraulically classified material. For example, the products from the intermediate spigots of a multiple-spigot classifier are comparatively free of all finer material, and the coal is substantially coarser than the accompanying heavy impurity.

Table XXI. Float-and-Sink-Separations at 1.60 Sp Gr on Products from Spiral Test of Roslyn Coal

Product	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative Weight, Pct	Cumulative Ash,[a] Pct
Washed coal	Under 1.60	96.4	9.6	96.4	9.6
Weight, 72.6 pct	Over 1.60	3.6	59.4	100.0	11.4
Middling	Under 1.60	69.2	11.7	69.2	11.7
Weight, 12.7 pct	Over 1.60	30.8	68.4	100.0	29.2
Refuse	Under 1.60	27.6	11.3	27.6	11.3
Weight, 14.7 pct	Over 1.60	72.4	76.7	100.0	58.6

[a] Moisture-free basis.

Table XXII. Screen Analyses of Specific Gravity Fractions of Spiral Products from Roslyn Coal, Pct

Product	Weight, Pct	Screen Size, Mesh 8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	72.6						
Under 1.60	96.4	52.5	26.5	10.6	6.2	2.8	1.4
Over 1.60	3.6	32.7	7.5	3.2	9.9	19.8	26.9
Middling	12.7						
Under 1.60	69.2	8.0	41.1	31.4	13.8	3.9	1.8
Over 1.60	30.8	72.1	13.1	3.3	4.0	3.7	3.8
Refuse	14.7						
Under 1.60	27.6	0.0	9.9	43.7	34.1	10.0	2.3
Over 1.60	72.4	13.9	35.4	26.1	16.7	5.0	2.9

In an effort to demonstrate that a hydraulically classified feed would permit better spiral operation, a sample of Roslyn coal was treated in a single-spigot classifier adjusted to remove as a spigot product only the coarse impurity. The overflow product from the classifier was then pumped through a cyclone to eliminate part of the material finer than 200-mesh. The resulting roughly classified product formed the feed for a spiral test in which a two-product separation was made.

The results of this test are compared with those for a test on a natural feed in table XXIV. With the spiral adjusted to give a refuse product of 60 pct ash, corresponding to the best refuse obtainable in tests on natural coal, it was possible to obtain a washed coal containing 11.7 pct ash instead of 14.3 pct. The washed coal from the classified feed contained only one half as much heavy impurity as that in the washed product obtained from a natural feed. The screen analyses of the heavy impurity contaminating the washed coal in these two tests demonstrate that classification of the feed minimized the amount of coarse impurity entering the washed coal. The presence of 26 pct of heavy impurity finer than 200-mesh in the washed coal from the classified feed is attributable largely to the fact that the clay associated with Roslyn coal slimes so readily. Had this test been made on a coal containing impurity that was more stable in water, the improvement possible through classification would have been more pronounced.

These rather fragmentary results appear to bear out the contention that preclassification of spiral feed would be beneficial. It must be realized also that silt bank material or the underflow product of a thickener operated as a hydroseparator such as currently are being treated in the spiral[8] constitute semiclassified feeds.

Table XXIII. Ash Content, Yield, and Efficiency by Particle Size for a Two-Product Separation of Roslyn Coal

Size, Mesh	Weight, Pct	Ash[a] Content, Pct: Feed	Ash Content: Washed Coal	Ash Content: Refuse	Yield, Pct: Washed Coal	Yield, Pct: Float Coal	Efficiency, Pct
8 to 14	42.5	17.7	15.4	80.4	96.7	98.1	96.6
14 to 28	27.0	20.4	11.1	70.7	84.4	86.5	97.6
28 to 48	15.0	22.2	9.9	50.7	69.3	81.4	85.1
48 to 100	9.1	24.2	11.9	47.2	65.2	84.3	77.3
100 to 200	3.9	27.2	20.7	46.9	76.2	95.1	80.1
Under 200	2.5	38.5	35.2	58.8	85.6	97.0	80.2
Composite	100.0	20.6	14.6	58.6	85.3	92.8	91.9

[a] Moisture-free basis.

Table XXIV. Comparison of Spiral Tests Made on Natural and Classified Roslyn Coal

	Type of Feed: Natural	Type of Feed: Classified
Ash content of feed, pct	21.8	16.8
Ash content of washed coal, pct	14.3	11.7
Ash content of refuse, pct	59.3	60.2
Yield of washed coal, pct	83.3	89.4
Sink 1.60 specific gravity in washed coal, pct	7.7	3.5
Screen analysis of impurity in washed coal, pct		
8 to 14	64.0	12.4
14 to 28	9.0	22.9
28 to 48	2.7	9.6
48 to 100	4.2	11.6
100 to 200	7.5	17.5
Under 200	12.6	26.0

Acknowledgments

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Petalite—A New Commercial Mineral

by John D. Clark

ONE'S lifetime does not usually offer the opportunity to observe and be a part of the commercial development of an entirely new material. Petalite has been known for over a century, but at first it was presumed rare. However, it was found to be so abundant that it has become a new and extremely useful industrial tool.

Mineralogy

Petalite is a lithium-aluminum-silicate mineral. It was from petalite that J. A. Arfwedson first discovered lithium in 1818. The theoretical composition of $Li_2O \cdot Al_2O_3 \cdot 8SiO_2$ is equivalent to 78.4 pct silica, 16.7 pct alumina and 4.9 pct lithia. The mineral in its known commercial occurrence is colorless to white and rarely gray, transluscent and rarely transparent. It is brittle, has a vitreous luster, and occurs usually in cleavable masses (fig. 1). It is classified as a di-silicate, but is similar in composition to the metasilicate spodumene, $Li_2O \cdot Al_2O_3 \cdot 4SiO_2$ (table I). However, their specific gravities are quite different (3.13 to 3.20 for spodumene, 2.39 to 2.46 for petalite).

Occurrence

Petalite specimens have been reported in many localities; however, there are only three known countries having occurrences of commercial significance. The classic locality is Sweden's Utö Mines, which, although reportedly still containing considerable petalite, have been closed and are not expected to be operated in the near future. At Varuträsk, in northern Sweden, petalite carrying 4 pct lithium oxide is the main ore. In 1946 there was reported about 1000 tons on hand. However, the iron content was 0.60 pct too high for other than chemical use.

Petalite has been reported in western Australia. At the Cottesloe mine, where small quantities have been reported available, it is produced in conjunction with feldspar. Petalite occurs in considerable quantity in the feldspar quarry at Londonderry, and it was estimated[1] in 1943 that over 3700 tons might be available.

The petalite in South West Africa was first discovered by F. J. Jooste, a lawyer by profession, in 1938. The following year he discovered a much larger deposit, and in 1948 another important discovery was made. These deposits occur near Karibib and constitute the only known commercial deposits. Their full extent has not yet been determined, but huge massive blocks of petalite of the purest grade, running into many thousands of tons, can be seen in all these deposits. To date, 60,000 tons of high grade, low iron containing ore has been blocked out in the major bodies. Mining for years to come will be confined to simple quarrying. Selection from accessory minerals has proved

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Fig. 1—Specimen of translucent petalite from South West Africa. The evidence of cleavage shown is not generally so prevalent. Massive petalite is readily mineable free of all accessory minerals.

to be no problem. Transportation to dockside is more than adequate.

The mineral occurs in quartz blows, generally surrounded by granite and pegmatite bodies parallel to, or in line with, the lepidolite bodies, with amblygonite and beryl on a much smaller scale. Maps or charts of the deposits, unfortunately, are not yet available.

It is interesting to note that petalite has the appearance and characteristic mode of occurrence of pegmatitic quartz. It is not surprising, therefore, that for a great number of years many geologists of high repute, mining engineers and inspectors of mines, are reported to have walked over and even sat on these deposits without noticing the existence of this mineral. This is understandable because the cleavage of petalite, called "perfect" in handbooks, has been observed to be poor to fair in specimens examined.

In October 1945 at the AIME meeting at Franklin Institute, Philadelphia, I presented a paper[2] entitled "Lepidolite—Its Occurrences and Uses." Among the world's known lepidolite occurrences, I singled out the deposits of South West Africa with particular detail. I described how the ore bodies dipped with the pegmatites with which they are associated with white pegmatitic quartz. It is believed that this reported pegmatitic quartz may have been wholly petalite.

This paper also stated that "recent word from the mines is that petalite has been described as most abundant." No further details of origin were available at that time. The first shipment of several tons arrived in Philadelphia during the spring of 1945.

Research and Development

By early 1947 the advent of jet and rocket engines had stimulated great interest in the discovery and application of heat-resisting ceramic materials. These were to withstand rapidly repeated heat shock up to and exceeding 1400°C (2550°F). Many trial bodies, which at first seemed promising, failed during endurance tests. However, these same bodies proved quite satisfactory for less technical applications, especially when operating temperatures were at 1300°C (2372°F) and lower, and when the rate of temperature change was measured in minutes instead of seconds. Thus, Floyd Hummel of Pennsylvania State College, who had been interested in the lithia-alumina-silica system, undertook further private investigations of this promising "tangent" early in 1947. Being first interested in the structural relationship between spodumene and cordierite, he extended his scope of interest to petalite. A study of the lithia-alumina-silica system was made by R. A. Hatch.[3]

A paper by R. Roy, D. M. Roy, and E. F. Osborn,[4] broadened the fundamental knowledge of the lithia-alumina-silica system. The molecular formula of petalite is in doubt according to Roy. Evidence indicates that the theoretical ratio of lithia-alumina-silica may be other than 1:1:8 as generally quoted in all handbooks.

The preliminary report of Hummel's[5] development of natural petalite ceramic bodies indicated that natural petalite undergoes a dissociation into beta spodumene and amorphous silica upon heating to 1100°C. This dissociation is irreversible, which is ideal to produce thermal shock-resisting ceramic bodies. Since this report, many investigations have been conducted portending a wide scope of commercial applications.

Thermal Expansion of Natural Petalite

Petalite begins to dissociate into beta spodumene and a siliceous glass around 1000°C, resulting in a linear coefficient of expansion of 19×10^{-7} cm per cm

per °C, with movement coming in the early stages of heating. At this point, the mineral is apparently undissociated. Firing to 1100°C for an hour brings about an unusual change in that the expansion is now a straight line type and about the same magnitude as that of fused silica. Heating to 1250°C lowers the linear expansion below that of fused silica.

Theoretical Considerations

The fact that lithium resembles certain members of the alkaline earth group has been reported on numerous occasions, as has the anomalous behavior of silicate glasses containing the ions Li^+, Be^{++}, and Mg^{++}. These ions, smaller than ions such as Na^+, K^+, Ca^{++}, Sr^{++}, Ba^{++}, and Pb^{++}, give rise to "contracted" glasses, that is, glasses that have a higher density than would be expected from theoretical calculations. They also have higher densities than the crystalline phases of the same compositions. As a glass cools from its molten condition, its density apparently depends on the size of the cations is has to accommodate. Evidence of this is indicated by studies of cordierite (a magnesium-aluminum-silicate) and spodumene as well as petalite.

Therefore, in practice, low expansion petalite ceramics are produced in a porous sintered form, taking full advantage of the minimum stress indicated by the combined low expansion of the silica residue and beta spodumene. Such bodies have a relatively long firing range.

To produce dense ceramics of low porosity, as well as expansion, advantage is taken of the admixtures, such as zircon, which act as inert low expansion fillers and provide means by which a maximum of sintered petalite can be retained out of solution. This also tends to lengthen the firing range of such bodies.

Commercial Applications

Past applications of so-called "heat shock" ceramic bodies were limited to a few conventional types such as alumina, mullite, zircon, and cordierite. Under acute conditions it is necessary on occasion to resort to fused silica articles.

Commercial alumina or corundum bodies have linear coefficient of expansion in the range of 70 to 85 mullite and zircon from 40 to 45, and cordierite from 15 to 30 (all $x\ 10^{-7}$ cm per cm per °C). Fused silica is in the range of 5 to $8x10^{-7}$. Porous or dense natural petalite bodies are adaptable to all ceramic forming practices. They can be produced having a low positive thermal expansion equivalent to fused silica, down to having to a low thermal contraction. Within this range, bodies can be formulated having zero expansion from 0 to 700°C. Such bodies require precise firing conditions and composition such that at least partial synthesis is necessary. Interferometer tests are used on all tests to eliminate the margin of error in conventional fused silica apparatus.

Table I Composition of Petalite

	1	2	3	4	5	6	7	8
SiO_2	78.68	76.19	76.16	76.50	76.99	75.30	62.91	51.74
Al_2O_3	16.62	16.48	17.24	17.50	16.25	17.20	28.42	27.50
Fe_2O_3	0.09	0.21	0.10	0.18	0.05	0.06	0.53	0.08
FeO	nil	nil	[b]	[b]	[b]	[b]	[b]	[b]
Mn_2O_3	0.004	trace	trace	[b]	[b]	[b]	[b]	0.15
MgO	nil	0.54	0.24	[b]	[b]	[b]	0.13	[b]
CaO	nil	nil	0.21	0.10	[b]	[b]	0.11	[b]
Li_2O	4.13	3.72	3.62	4.50	4.41	3.94	6.78	4.30
Na_2O	0.08	0.36	[b]	0.16	[b]	[b]	0.46	0.94
K_2O	nil	0.18	0.32[a]	0.39	0.48[a]	0.20[a]	0.69	7.40
Rb_2O	[b]	[b]	[b]	[b]	[b]	[b]	[b]	2.20
H_2O^+	0.01	1.04	[b]	[b]	[b]	[b]	[b]	[b]
H_2O^-	nil	1.22	0.24	0.80	0.25	[b]	0.28	[b]
F	[b]	[b]	[b]	0.11	[b]	[b]	[b]	7.20

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2. Petalite, colorless milky, Cottesloe, Western Australia by Gov. Lab.
3. Petalite, colorless milky, Karibib, S.W. Africa by Foote Mineral Co.
4. Petalite, colorless milky, Karibib, S.W. Africa by Foote Mineral Co.
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[a] All "other alkalies" reported as K_2O.
[b] Not determined.

The commercial range of applications extends up to 1200°C (2192°F), which is as high as cordierite and higher than the recommended maximum for fused silica. Advantage over transluscent fused silica should be evident in the range 900°C to 1200°C, where the glass product is susceptible to more or less rapid devitrification. Thus, petalite ceramics constitute a bright, new tool on the industrial horizon. They have been successfully tested for many uses including pyrometric and combustion tubes and boats, refractory trays, saggers, batts and furniture, burner tips and blocks. They are even adaptable to oven and top-of-stove ware.

The electrical properties of such bodies are equivalent to L-3 type steatite porcelain, which has a notably poor resistance to heat shock. Spark plug insulators are being investigated.

The molecular formula of natural petalite closely approximates the oxide ratio of some ceramic glazes. Simple modification results in glazes suitable for hotel china, chemical and electrical porcelain, and sanitary ware.

Petalite has a lower alumina-lithia weight ratio than spodumene (slightly) or lepidolite, which makes it a preferred lithia source in low alumina-containing porcelain enamels and glasses.

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Tin Deposit of the Monserrat Mine, Bolivia

by Russell Gibson and F. S. Turneaure

Fig. 1—Location of Monserrat mine.

The tin deposit of Monserrat, Bolivia, consists of one major vein 1600 m in length. The ore is unusual because of the notable quantity of teallite, even though cassiterite is the principal tin mineral. The deposit, formed at shallow depth under a wide range in temperature, may be classed as xenothermal. Polished sections reveal a complex history of replacement, with low-temperature minerals deposited before high-temperature minerals.

THE Monserrat mine[1, 2] of the Compañia Minera Monserrat is located 11 km east of Callipampa, a station on the Antofagasta and Bolivia Railway some 55 km south of Oruro. The mine lies in a group of low hills on the east slope of a broad valley of northerly trend. The valley is separated from the altiplano to the west by a prominent ridge that rises several hundred meters above the surrounding country (fig. 1). Callipampa is situated a short distance west of the ridge, near the east margin of the altiplano. The elevation at Callipampa is 3700 m and at the mine 4100 m.

Monserrat is near the northeast limit of what may be called the Poopo-Pazña district, an area of northwesterly trend about 25 km in length and 15 km in width. The district includes the tin-silver prospects of Poopo to the north, the zinc-tin deposits of Salvador to the southwest, and the tin-tourmaline veins of Avicaya to the south. Along the prominent ridge already mentioned, which forms the western limit of the district, there are several tin and antimony prospects including the Trinacria mine, which is famous for its fine specimens of cylindrite.

Although the principal ore mineral is cassiterite, the tin deposit of Monserrat contains an unusual amount of teallite. Cassiterite occurs in part as a finely granular mixture with sulphides of iron and zinc and in part as needle tin of late hypogene age.

The regional structure is dominated by broad folds typical of the Andes of Central Bolivia and similar to those of the Llallagua district. The area adjacent to the Monserrat mine is underlain by thin-bedded shale somewhat variable in color but predominantly dark grey to black. The beds strike northwest and dip at low angles southwest, forming part of the northeast limb of a major syncline. A resistant sandstone formation, which underlies the black shale, crops out along a ridge east of the mine and also accounts for the high ridge to the west that was previously mentioned.

No igneous rock of any kind is found at Monserrat. The nearest intrusive bodies known are the irregular dikes and masses of quartz porphyry at Avicaya about 10 km to the south.

Mine Workings

The Monserrat property includes one main vein, which can be followed on the surface for 1600 m, and two or three branch veins of distinctly minor importance. The principal haulage tunnel, referred to as the San Carlos adit or level III, extends northeasterly for 1700 m, following the main vein throughout most of this distance. Above the adit, there are

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Fig. 2—Galena (gn) replaced along cleavage by cassiterite (cs). X125.

Fig. 3—Bent grain of teallite (tl) replaced along cleavage by pyrite (py) and cassiterite (cs). X125.

Fig. 4—Curved teallite (tl) grains partly replaced by cassiterite (cs). X125.

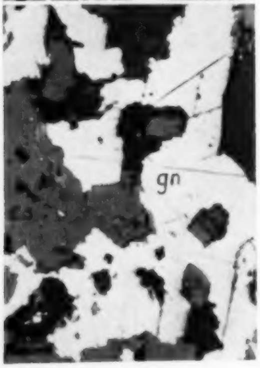


Fig. 5—Galena (gn) partly replaced by cassiterite (cs). Shape of cassiterite is, in part, influenced by cleavage of galena. Some grains are sections of cassiterite "needles." X125.

two principal horizons which are referred to as levels, although the drifts of each show minor differences in elevation. Because of the gentle slope of the surface, the two are considerably shorter than level III.

Several crosscuts have been extended from the drifts, especially in the hanging wall of the main vein. Development in depth has reached level VII, about 130 m below the adit level.

The Monserrat Vein

In view of the fact that the country rock is a soft thin-bedded shale, the persistence of the Monserrat vein is remarkable. The strike of the vein changes gradually from N 22°E at the south end to N 45°E at the north end and thus cuts across the bedding of the shale. The predominant dip is 75°SE, but in places it reaches 86° and in the northern section flattens to 70°.

The vein lacks a persistent gouge, and it probably does not correspond to a major fault. Rather, it is a complex lode with a considerable variation in detailed structure. In places, the lode consists of a single vein with rather sharp walls and a crudely banded sulphide filling; elsewhere it splits into several closely spaced but independent stringers. To some extent, the branching structure corresponds to low-grade sections of the ore-shoot.

The development of the main vein on level III can be considered typical. In the central part of the ore-shoot, the vein splits into two distinct branches separated by as much as 15 m of shale, the two maintaining their identity for 150 m on the strike. Two or three additional branch veins of distinctly minor importance have been prospected by short drifts, and a number of narrow sulphide stringers have been exposed in the exploration crosscuts.

The vein filling generally consists of massive and crudely banded sulphide more or less frozen to the walls. Locally, an open crustified filling is pronounced, especially where the sulphides give way to quartz. A brecciated structure is evident in one or two places on level III. The filling averages in width from 15 to 25 cm except where the vein splits into a number of stringers. Here the lode structure attains a width of 50 cm or more.

From the surface to level III, a maximum vertical distance of 135 m, the main vein is ore-bearing almost continuously throughout the 1600 m of length, although in the south half of the shoot there are some low-grade areas. At greater depth, the ore shoots become smaller, but the vein continues strong to level VII, 265 m below the outcrop.

Wall-rock alteration along the main veins is scarcely noticeable; as a rule it consists of only a feeble silicification. In a few places, alteration is more intense, and a siliceous shale with finely disseminated sulphide extends a few centimeters from the vein. The fragments of shale within the vein and the narrow bands between stringers are silicified and sericitized to some degree, but no marked change is visible in many exposures.

Vein Minerals

General Description: The description of the Monserrat ores is based upon an examination of levels II to V and on a study of representative specimens from levels II to VII.

The veins of Monserrat are of scientific interest because they contain teallite, an uncommon sulphide of tin and lead, abundant needle tin (acicular crystals of cassiterite), and wurtzite, the less common,

hexagonal form of zinc sulphide. In the tin veins of Carguaicollo,[3] described previously, teallite and wurtzite are abundant, and needle tin is present in small quantity, giving a mineral assemblage strikingly similar to that of Monserrat.

The dominant minerals of the veins are zinc sulphide (wurtzite and sphalerite), pyrite, cassiterite, teallite, and quartz. Less abundant are galena and carbonate. Arsenopyrite, marcasite, and bournonite are widespread but nowhere abundant. Stannite, chalcopyrite, and vivianite are of subordinate importance. Oxidized ore from upper workings collected from old dumps contains goethite, "limonite," jarosite, and kaolin minerals.

Banding is evident in many parts of the vein on all levels examined. The crude bands are commonly 1 to 5 cm wide, but examined closely these in turn are made up of bands from a fraction of a millimeter to 7 mm in width. The banding is emphasized in many places by the crustified filling and by the lines of vugs and druses. The vugs, rarely more than a few centimeters across, are lined by crystals of needle tin, pyrite, quartz, and sphalerite, and more rarely by wurtzite crystals and bladed vivianite. These features emphasize the fact that filling was the dominant method of mineralization at Monserrat.

Most of the ore is fine grained and some of it is exceedingly so, but the texture is erratic. The banding of the ore is due in part to variation in grain size. In some bands, the grains are 0.05 mm or less in greatest dimension, giving the ore a dense and massive appearance. In other adjacent or nearby bands, the grains are much larger, roughly a millimeter or two across; and lining the vugs, the grain size reaches 5 mm.

Cassiterite: Cassiterite is found in nearly every polished section examined. It occurs as shapeless grains, as veinlets in other minerals, and as prismatic grains of extremely small diameter (needle tin).

Granular cassiterite forms pseudomorphs after teallite and easily replaces galena, sphalerite, and wurtzite, especially along the cleavages of these minerals (fig. 2). Veinlets of cassiterite and galena cut sphalerite. Where large areas of teallite have been replaced wholesale by zinc sulphide, small curved residual shreds of teallite often remain in the field, and these remnants are often partly replaced by cassiterite (figs. 3 and 4).

Needle tin is widespread. It occurs as scattered, isolated euhedrons and in sheaf-like bundles, the latter especially well shown in association with late quartz in numerous vugs and druses. In hand specimen, the needles are straw-yellow; in thin section, colorless and pale yellow. Some grains are clear; others are turbid. The needles are commonly from 0.01 to 0.05 mm across and 0.20 to 0.50 mm long.

According to A. Hazard, the General Manager, needle tin is especially abundant in the northerly third of the vein above level III and also in the same general area between levels IV and V. But the needle tin is not confined to these places. It has been found in variable amounts from level I to level VII and through a lateral distance of 1300 m or more.

Like the granular cassiterite, needle tin is late in the sequence and replaces teallite, wurtzite, sphalerite, and galena (figs. 5 and 6). Needle tin is present in vugs perched on galena, teallite, and other minerals. Minerals associated with needle tin are the same as those associated with granular cassiterite, and there is no evidence that the needle tin belongs

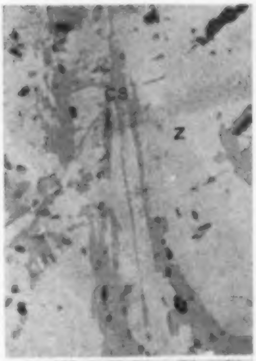


Fig. 6—Zinc sulphide, chiefly sphalerite (z) replaced by "needle tin" (cs). X125.

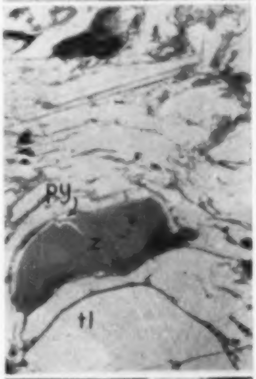


Fig. 7—Curved grains of teallite (tl) partly replaced by zinc sulphide (z). Later pyrite (py) replaces teallite and follows fracture in zinc sulphide. X125.

Fig. 8—Residual teallite (tl) laths in a field of zinc sulphide (z). X240.

Fig. 9—Early stage replacement of teallite (tl) by zinc sulphide (z) and pyrite (py). X125.

Fig. 10—Pyrite (py) pseudomorphs after teallite (tl). X125.

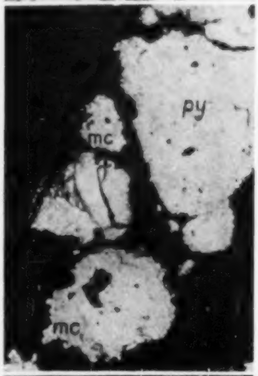


Fig. 11—Pyrite (py) corroded along rims by marcasite (mc). Nearly every pyrite grain is rimmed by more or less marcasite. X125.

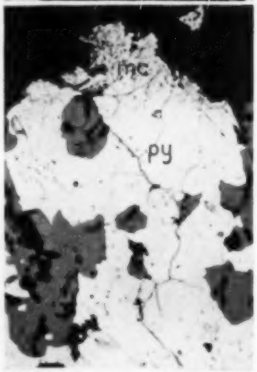


Fig. 12—Pyrite (py) partly replaced by marcasite (mc). X125.

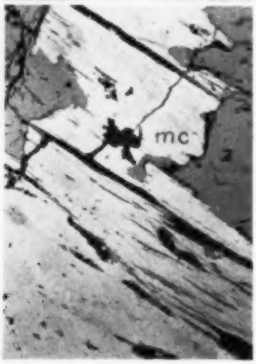


Fig. 13—Marcasite (mc) possibly pseudomorphous after teallite. X40.

to a different period of mineralization. There is no zone of supergene tin enrichment. Both the granular cassiterite and needle tin are late hypogene.

Teallite is clearly replaced by needle tin, granular cassiterite, and galena, but in sections showing this replacement there is no indication of supergene deposition of tin or other elements.

Teallite ($Pb\ SnS_2$): Teallite is a widespread mineral in the veins but is not everywhere abundant. In hand specimens, it is black, platy and shows good metallic luster. Commonly, grains are 0.05 to 0.10 mm long, but many are only one tenth this length. Rarely, a lath 1 or 2 mm long is seen in coarse-grained ore. Because of its black color and excellent cleavage, fine-grained teallite is difficult to distinguish from fine-grained galena in hand specimen. Because of the softness and flexibility of teallite, it records with special readiness and clarity minor deforming stresses that operated subsequent to its deposition. The elongated teallite grains at Monserrat are commonly curved or bent and with this often goes an opening up of the (bent) cleavage plates; some are deformed only slightly, whereas others are bent through 90° (fig. 3). This deformation is largely prior to and independent of any post-mineral faulting or brecciation of the ore.

Teallite has been replaced chiefly by zinc sulphide, (wurtzite and sphalerite), cassiterite, pyrite, and galena. The deformation of many of the grains, as well as the excellent cleavage, may have facilitated replacement (fig. 4). Some former curved grains of teallite are replaced entirely by cassiterite and galena, the surviving curvature helping to certify the pseudomorphous nature of the alteration (fig. 3). The sphalerite and wurtzite of some polished sections also show curved and distorted grain boundaries, interpreted as a texture inherited from early teallite now entirely removed. If this is correct, then teallite in the early geologic history of the vein was much more abundant than now.

Sphalerite and Wurtzite: Sphalerite and wurtzite are the most abundant metalliferous minerals and are among the most widespread. Nearly every section contains one or the other. To determine which is present, immersion tests or thin sections are necessary because the two minerals look alike and are seemingly anisotropic in polished section. Powder X-ray photographs were also used to confirm the identity of some grains. In the following account, "zinc sulphide" is used in referring to instances where the precise identity of the zinc sulphide has not been ascertained. Where its identity has been established, however, by any of the means mentioned, the precise name will be used. In 19 thin sections containing zinc sulphide, the relative percentages of sphalerite and wurtzite are 43 and 57.

Zinc sulphide is the dominant mineral in many parts of the veins and, in some places, gives the appearance of being present in relatively pure bands. Numerous polished sections show, however, that it is almost invariably intergrown with other minerals. Zinc sulphide began depositing early in the sequence, although probably later than teallite, and continued to form throughout most of the mineralizing period.

Crystals of both wurtzite and sphalerite have been identified in vugs. Both are dark brown to black and are indistinguishable merely on the basis of color. Most of the zinc sulphide, however, occurs in cleavable masses and bands as a dominant part of the ore. In thin sections, zinc sulphide is red,

reddish brown or shades of yellow, and grains of any of these colors may show good birefringence. Moreover, one color fades into another. Some of the wurtzite that shows good birefringence in thin section is bladed or radially fibrous. Some of this wurtzite replaces sphalerite. Polished sections show teallite partly or completely replaced by zinc sulphide, pseudomorphously in numerous places (fig. 7). Some of these pseudomorphs are adjacent to laths of teallite replaced by cassiterite. Seemingly there has been wholesale replacement of the tin sulphide mineral by zinc sulphide, and many fields of sphalerite or wurtzite show residual shreds, wisps or blades of teallite (fig. 8). Zinc sulphide is replaced in many sections by galena, cassiterite, pyrite, and marcasite, but there are some reversals in the sequence with pyrite and a very few with the others.

Pyrite: Pyrite is present in nearly every specimen examined but is not conspicuous in the ore. About one half the polished sections that contain pyrite show more or less marcasite. A little disseminated pyrite is present in the wall rock near veins. Pyrite commonly occurs in scattered grains or small patches of grains and in tiny veinlets and occurs less commonly as thin bands. Individual grains are usually 0.01 to 0.10 mm and rarely 0.30 to 0.70 mm in diameter. In about one third of the polished sections, grains were large enough to permit positive determination of anisotropism, commonly weak, but occasionally strong. Polarization colors are shades of blue and brown.

Pyrite was deposited throughout most of the vein-forming period, although the greater part of it was deposited later than the abundant minerals, sphalerite, cassiterite, teallite, and quartz. It is found as veinlets along the cleavage of some minerals or cutting across aggregates of grains and as pseudomorphs after teallite (figs. 9 and 10). Residual patches of early pyrite are seen in zinc sulphide clearly not related to any cleavage. Some of the veinlets and residual grains are partly replaced by marcasite (figs. 11 and 12). Pyrite cannot be separated into early and late phases on the basis of size of grain or anisotropism.

Marcasite: Marcasite is of minor importance quantitatively, but about two thirds of the specimens contain at least a little. Most of it was observed only in polished sections. Marcasite occurs commonly as veinlets, 0.005 to 0.01 mm wide, cutting and replacing other minerals and to a less extent as clumps of grains, the individuals of which range up to 0.15 mm in diameter.

Marcasite is later than pyrite with which it is most closely associated. It also veins zinc sulphide, cassiterite, galena, and carbonate and occasionally forms lath-like or lamellar pseudomorphs about 0.50 mm long after teallite (fig. 13). Marcasite is clearly late in the sequence.

Galena: Galena is present in about one third of the polished sections examined but is abundant in very few. It is not conspicuous in the hand specimens and would be overlooked in most of them, especially since it is the same color as teallite. All polished sections that show galena also contain the lead-bearing mineral teallite.

Galena occurs in irregular small grains, in veinlets in other minerals, and in pseudomorphs after teallite. Many curved or wisp-like grains originally composed of teallite are now converted wholly to galena or galena and cassiterite, although residuals of teallite remain in some (figs. 14 and 15). Veinlets

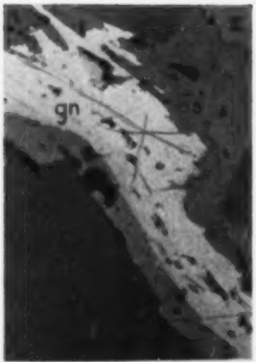


Fig. 14—Curved teallite completely replaced by galena (gn) and cassiterite (cs). X125.

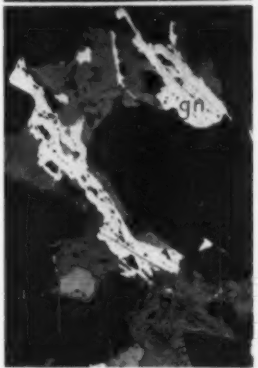


Fig. 15—Curved teallite completely replaced by galena (gn) and cassiterite (cs). Note early subhedral quartz (q). X40.

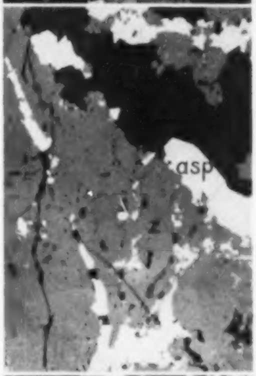


Fig. 16—Curved grains of arsenopyrite (asp) suggesting control by early teallite. Arsenopyrite is corroded by zinc sulphide (z). X40.

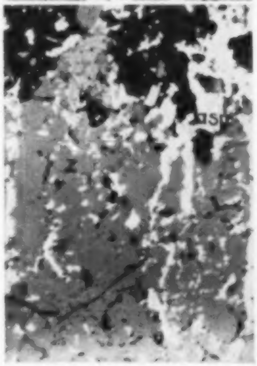


Fig. 17—Strings of grains of arsenopyrite (asp) in zinc sulphide (z) and quartz (q). Arsenopyrite is partly replaced by zinc sulphide. X125.

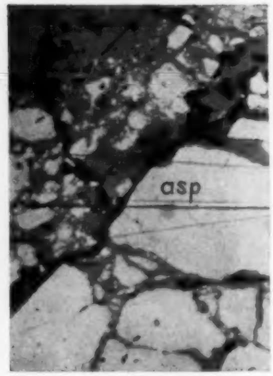
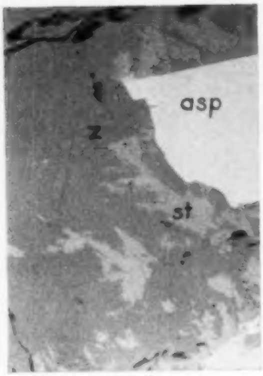


Fig. 18 (left)—Broken arsenopyrite (asp) partly replaced by zinc sulphide (z). X125.

Fig. 19 (right)—Zinc sulphide (z) replaced along cleavage by stannite (st). (Exsolved stannite?) X125.



of galena cut sphalerite and wurtzite along cleavage. Residual grains of teallite in a field of later zinc sulphide are replaced by galena or galena and cassiterite. In some places galena and cassiterite are seemingly contemporaneous; elsewhere, cassiterite is slightly later than galena as shown by veinlets along galena cleavage.

When teallite became unstable in the later hypogene solutions, some grains were converted to galena and cassiterite, but many were replaced almost entirely by cassiterite. The alteration thus involves an introduction of tin by the hydrothermal solutions and does not represent simply the recombination of the tin present in the teallite. At this stage, the solutions may have been of higher intensity so that cassiterite was readily deposited both as a replacement of teallite and as a coating of needle tin, whereas much of the lead sulphide was swept upward to cooler regions.

Arsenopyrite: Arsenopyrite, a minor but widespread constituent of the ore, is present in about two thirds of the polished sections, but in only one does it amount to more than 5 pct. The grain size commonly ranges from 0.003 to 0.20 mm and exceedingly few grains are larger than 0.50 mm in diameter.

Arsenopyrite occurs in swarms and strings of small grains, in isolated euhedrons, and to a less extent as veins in other minerals. It is very commonly present at the grain boundaries of quartz with sphalerite or wurtzite (fig. 16) or of pyrite with zinc sulphide. It replaces bent teallite grains and galena that was probably derived from teallite; it also occupies intergranular spaces and forms veins in areas of zinc sulphide (fig. 17). Yet curved plates of arsenopyrite that were probably derived from teallite are corroded by zinc sulphide (fig. 16), and in one area broken arsenopyrite grains are surrounded by zinc sulphide (fig. 18). Although arsenopyrite is commonly later than most of the abundant minerals, it shows some overlap with zinc sulphide.

Stannite and Chalcopyrite: A few small grains of stannite, 0.01 to 0.05 mm in diameter, were seen in polished sections. The stannite is almost invariably in sphalerite or wurtzite and may represent exsolved material from the zinc sulphide similar to chalcopyrite in numerous other occurrences of sphalerite (fig. 19). However, Monserrat is notably lean in copper, and only two sections, both cut from the same hand specimen, show any chalcopyrite. In both sections the amount is exceedingly small. The chalcopyrite occurs as veinlets in bournonite and stannite.

Bournonite: Bournonite is present in trivial amounts, usually a small fraction of 1 pct, in nearly one half the polished sections examined. The grains are irregular in shape, except where replacing teallite, galena, or sphalerite along cleavage, and are commonly 0.10 to 0.50 mm in size. They usually show multiple twinning and strong birefringence. Bournonite's place in the sequence is not certain, but it is probably late. It clearly replaces the abovementioned minerals and is found at the borders of galena, sphalerite, quartz, and siderite. It contains residual wisps of teallite. It is replaced by chalcopyrite.

Gangue: The gangue minerals, chiefly quartz and carbonate with small amounts of other minerals, are subordinate to the metallic minerals and are not conspicuous.

Quartz, the most abundant and most widespread of the gangue minerals, is present in nearly every thin and polished section. It occurs in many textures and its deposition continued over a long period of time. In one half the polished sections, the quartz appears as somewhat corroded euhedrons that range from 0.02 to 0.50 mm in length. These are surrounded by teallite or by aggregates of later minerals formed largely by the replacement of teallite (fig. 14). In places, the ragged residual grains are almost completely replaced by metallic minerals.

Quartz occurs less commonly as mosaic patches of ordinary vein quartz with wavy extinction. In some of these patches, hundreds of tiny grains of arsenopyrite, pyrite, and sphalerite are present in a field only 2 mm in diameter. Quartz is present also in vugs with needle tin and pyrite crystals and in late veinlets accompanied by marcasite, one of the last minerals to be deposited.

Carbonate is present in nearly one half the thin and polished sections, but invariably in small amounts. It rarely can be seen in large enough grains to be identified in hand specimen. When tested by immersion or microchemistry, several of these proved to be siderite. Under the microscope carbonate is seen in veinlets cutting zinc sulphide, galena, cassiterite and other minerals, and in irregular grains interstitial to quartz, pyrite, and wurtzite. It is present in vugs, also, as a late mineral. In places, carbonate is replaced by late pyrite.

In several specimens a little vivianite was observed in clusters of radiated or divergent blue

crystals in vugs. Commonly, they are perched on siderite, but to a minor extent the siderite is later than the vivianite. The crystals range from 4 to 12 mm in length. A little sericite is present in the vein quartz as well as in the wall rock near the veins.

Summary of Sequence: Although some overlap in deposition is indicated by the textures, four principal episodes may be distinguished in the formation of the ore: (1) Early deposition of vein quartz, (2) formation of teallite, (3) widespread replacement of teallite by abundant sphalerite and wurtzite, and by arsenopyrite, galena, cassiterite, pyrite, and marcasite, and (4) continued deposition of late gangue, including both quartz and siderite (fig. 20).

Origin

A number of features show that the ores of Monserrat were formed at moderate to shallow depth. Most of the ore is fine grained and some of it is exceedingly fine grained. Crustified structure with drusy cavities and fine banding are common. The view that the deposit is of shallow type also receives support in its geological setting, that is, in the narrow but persistent vein enclosed in soft, unaltered shale.

The mineralogy of the ore points to deposition over an unusual range in temperature. Many of the minerals, such as pyrite, sphalerite, galena, chalcopyrite, quartz, and sericite, are stable over a wide range of conditions and give no clue to the temperature at which the deposit was formed. Cassiterite and arsenopyrite, however, normally are associated with high intensity minerals and point to high temperature. According to Gordon Smith's[4] recent experimental work, cassiterite can be crystallized from aqueous solutions of sodium stannate and in the laboratory from room temperature up to at least 450°C. Marcasite and wurtzite, on the other hand, are normally regarded as low-temperature minerals. The stability relations of teallite are not accurately known, but its associations in many deposits in Bolivia indicate that it is probably deposited under medium to low intensity conditions. The presence of all these minerals of extreme temperature range at Monserrat is an expression of telescoping.

In mineralogy, sequence of deposition, and origin, Monserrat offers interesting comparisons with Llallagua[5] and particularly with Carguaicollo.[5]

At Carguaicollo, teallite and franckeite are succeeded by cassiterite. At Llallagua, franckeite is followed by wolframite, pyrrhotite, and arsenopyrite. At Monserrat, teallite is followed by sphalerite and this in turn by arsenopyrite and cassiterite. In all three localities the tin-bearing sulphides are relatively early in the sequence, and these minerals, probably to be regarded as indicative of moderate to low temperature, are followed by minerals commonly regarded as minerals of high temperature significance. The sequence is thus opposed to the conventional view that deposition proceeds on a descending temperature scale. But it supports Graton's[6, 7] hypothesis that, at least during certain periods of the mineralizing process, low-temperature minerals are followed by those of higher temperature. Graton concludes that this must be true for at least the early stages of the mineral-depositing process, and we think that this is confirmed at Carguaicollo and Monserrat. At Llallagua, however, the temperature changes were more complex; early tourmaline, cassiterite and bismuthinite (high temperature) are followed by franckeite (low temperature) and this in turn by wolframite, pyrrhotite and arsenopyrite (high temperature). Sales and Meyer,[8] in their recent paper on the vein formation at Butte, Montana, suggested that mineralization took place there, also, on a rising intensity scale. "The zonal growth of the Butte district to its present dimensions strongly urges the proposition that at a given reference point in the vein system the intensity factor increased, on the average, during the growth of the veins." . . . "We must entertain the possibility of a slow average rise in temperature and chemical potential, two important components of the composite intensity factor, until the end of effective hydrothermal supply." Hart,[9] also pointed out at Butte that deposition may occur on a rising intensity scale. He mentions specifically sphalerite followed by copper-iron minerals.

Lacy[10] noted at one place at Cerro de Pasco that high-temperature minerals followed low-temperature minerals. "Along the solution channelway, heat was introduced faster than it could be dissipated by conduction, so, at one place along this channel, there were rising thermal conditions. Accompanying this rise in thermal gradient, the nature of the iron mineralogy changed from pyrite to pyrrhotite and magnetite, i.e., minerals usually assumed to indicate higher temperature were formed."

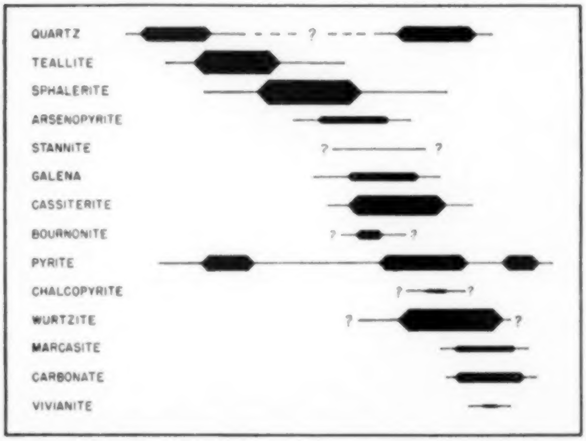


Fig. 20—Graphical representation of mineral sequence.

It is recognized that this argument is based heavily on the influence of temperature as the effective cause of deposition of ore minerals. It is granted that this is only one of several factors that operate to cause deposition. As pointed out in the next paragraph, a change in the alkaline-acid character of the solutions may have been an important contributing cause in the deposition of cassiterite.

The assemblage of minerals at Monserrat is, in the main, similar to that in other districts where an alkaline character of the ore-depositing fluid has been assumed. Smith reports that cassiterite is stable in hot alkaline solutions and can be crystallized from such solutions. He concluded that "in nature, if tin is transported as alkali stannate dissolved in aqueous solutions, a decrease in alkalinity of the solutions may precipitate cassiterite (and) if tin is transported from magmas as alkali thiostannate dissolved in aqueous solutions, a decrease in alkalinity and a decrease in the sulphide ion concentration due to reaction with vein wall rocks would both favor precipitation of cassiterite."[11] The abundance of wurtzite and the widespread development of marcasite at Monserrat, minerals known to crystallize in the laboratory[12] from acid solutions, suggest that the solutions may have decreased in alkalinity as deposition went on. This may have been a contributing cause in the deposition of cassiterite at Monserrat and perhaps at Carguaicollo. Anderson,[13] in his work at the Last Chance and Hornsilver mines in Idaho, notes that wurtzite is a late mineral in his "early assemblage mineral succession" and concludes that the deposition of wurtzite was favored by a decline in the temperature of the solution or by an increase in the acidity.

Decrease in alkalinity may not be the cause of deposition of cassiterite at Llallagua and Oruro. At these places cassiterite was deposited long before the minerals that suggest an approach toward acid solutions, i.e., marcasite, alunite, dickite, and kaolinite. Moreover, the deposition of sulphide in solution can scarcely be a major factor at Llallagua and Oruro, for at these localities the bulk of the sulphides follows the cassiterite.

Chace[14] concluded in his study of the Oruro tin-silver veins that the solutions that deposited those ores were probably alkaline at the beginning of the mineralization period but gradually became neutral, then slightly acid, and finally distinctly acid. His conclusion is strengthened by the fact that kaolinite, dickite, and alunite are present in the Oruro veins. Alunite was reported in the Monserrat ores by Ahlfeld,[15] but none was observed in the present study.

Monserrat, like Carguaicollo and Llallagua, does not fit well into Lindgren's genetic classification of ore deposits. In texture the deposits resemble epithermal ores and were probably formed in a shallow environment at low pressure. They correspond in these respects to epithermal deposits. But the temperature, in the middle period of mineralization at Monserrat and Carguaicollo at least, became abnormally high for this environment, and high-temperature minerals not characteristic of epithermal ores were deposited following low-temperature minerals.

The combination of high temperature and shallow depth would promote rapid deposition accompanied by telescoping. In a nonvolcanic environment such as the shale terrane at Monserrat, heating of the vein walls by the solutions themselves is of major importance, and this would favor deposition of minerals on an ascending temperature scale. Moreover, the change in the character of the solutions from alkaline to acid, as postulated at Monserrat, is most likely to develop in a shallow environment at low pressure.[16, 17] Thus the complex textures and widespread replacement so characteristic of the ore are readily explained by the rapid shifts in temperature, the low pressure, and the change in the character of the solution from alkaline to acid.

We have used Buddington's term, xenothermal, to classify the great tin deposit at Llallagua, which has an abundance of high and low-temperature minerals. The same term can be extended to Carguaicollo and Monserrat, even though these ores exhibit a milder degree of telescoping than do those of Llallagua.

Acknowledgments

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aime NEWS

Roy O'Brien Named Western Secretary, Mining Branch

Creation of a new office to serve western members of AIME was revealed by the Board of Directors at Salt Lake City with the announcement that Roy E. O'Brien has been appointed Western Secretary of the Mining Branch.

Mr. O'Brien's territory will embrace Montana, Colorado, Wyoming, New Mexico and all states west. He expects to spend much of his time in the field, coordinating the activities of Local Sections and Student Chapters, arranging divisional meetings, pushing the membership drive, getting material for Institute publications, and solving Institute problems in general.

Mr. O'Brien was born in Colorado, worked in the Phelps-Dodge

Roy E. O'Brien

coal mines, served in the Army during World War I, and then graduated from the University of Colorado in 1923. For nine years he was a shift boss and general foreman for Anaconda in Cananea, Mexico, and then, in 1935, went to the Mountain City Copper Co. in Rio Tinto, Nev. During a fourteen-year period he served the company as shift boss, mine foreman, and mine superintendent. When Mountain City was liquidated in 1949 he went with Anaconda's phosphate division in Conda, Idaho, and more recently has been with Roger B. Pierce, as a consultant on mine appraisals.

MBD Holds Technical, Business, Social Gatherings at Salt Lake

Nearly 175 members of the Minerals Beneficiation Division, AIME, gathered in Salt Lake City at the fifth general meeting on Aug. 31 and Sept. 1, to hear eight technical papers, transact Division business, and partake of the famed MBD Scotch Breakfast.

Early-rising members of the Division began the proceedings on the first day with the Scotch Breakfast, at which Scotch in the orange juice, porridge, eggs, coffee, and water was *de rigeur*.

That afternoon, the Division's business meeting considered various topics, among them technical papers, the annual meeting program, and MBD's share of the AIME membership drive. Program Chairman Raymond Byler introduced the idea that too many technical sessions were being held at annual meetings. A discussion of this point revealed the general feeling that MBD sessions should be reduced to six instead of eight, thus leaving two afternoons free at the meeting. Mr. Byler was consequently instructed to do as he saw fit, bearing in mind the various points raised during the discussion.

Taking his cue from the Mining Branch Council meeting the previous day, S. J. Swainson brought up the matter of restricting the acceptance of technical papers to maintain the high calibre of Institute literature. The MBD was generally in agreement with this suggestion. Mr. Swainson also reported that MBD was allotted 185 pages of Transactions for 1951.

Preliminary plans for an organized membership drive were mentioned by James Barr, Jr., who reported that 11.3 pct of AIME membership, or approximately 2814, are affiliated with MBD. This makes MBD the third largest division, outnumbered only by the Mining, Geology and Geophysics Division and the Petroleum Division. MBD plans to make a concerted effort to obtain new members from among the graduates in mineral dressing.

Secretary-Treasurer Will Mitchell proferred the information that the BMD treasury now holds over $2000. The Secretary's company, Allis-Chalmers Mfg. Co., was given due credit for an assist on the financial front through its efforts in behalf of MBD.

Speakers at the luncheon on Friday included President D. H. McLaughlin, Past-President L. E. Young, and Dean Carl J. Christensen of the University of Utah. Roy E. O'Brien, new Western Secretary of the Mining Branch, was introduced by Chairman Grover Holt.

Nominations for MBD officers for 1951 were announced:

Chairman, Raymond E. Byler; **Associate Chairman,** Edwin H. Crabtree; **Regional Vice Chairmen,** Donald W. Scott and S. D. Michaelson; **Secretary-Treasurer,** Will Mitchell, Jr.

COMMITTEE CHAIRMEN:— **Papers and Publications,** M. D. Hassialis; **Membership,** S. E. Erickson; **Education,** H. Rush Spedden; **Program,** E. H. Crabtree; **Symposium,** Norman Weiss; **Concentration,** J. C. Lokken; **Materials Handling,** W. B. Stephenson; **Crushing and Grinding,** R. J. Russell; **Solids-Fluids Separation,** J. L. Weaver; **Operating Control,** C. M. Marquardt; **Solution and Precipitation,** James H. Jacobs; **Pyrolisis and Agglomeration,** F. M. Hamilton; **Richards Award,** S. J. Swainson and T. B. Counselman.

Bowling League Aids Local Section

Jack Ehrhorn of the Utah Section of AIME is the daddy of a bowling league which has done a great deal toward drawing engineers together for the exchange of ideas and broadening of acquaintances. Started four years ago, the league now boasts 12 teams. Only AIME members and good prospects can participate.

Los Angeles Meeting

October 12-13, 1950
Headquarters, Elks Club

The Industrial Minerals Division, as part of the Mining Branch, is meeting at the Elks Club, 607 South Park View, at the same time as the Petroleum Branch (Oct. 12-13) and the Metals Branch (Oct. 12). The following program is sponsored by the Mining Branch and includes papers of both the Industrial Minerals Division and the mining and milling groups.

FRIDAY, OCTOBER 13

9:00 a.m.—Registration of all members and guests and sale of preprints. Main Entrance Hall, Elks Club.

TECHNICAL SESSION, 9:30 A.M.

Conrad Thomas and C. R. King, Co-chairmen. El Venado Room, Elks Club.

Opening Remarks. By George D. Dub, Western Vice-chairman, Industrial Minerals Division; Chairman, Southern California Section.

California's Mineral Position in a Changing Industrial World. By Olaf P. Jenkins, chief, State Division of Mines.

Production of Iodine in the United States. By Jack E. Ryan, Research Dept., American Potash & Chemical Corp.

Searles Lake and Operations of the American Potash & Chemical Corp. at Trona, California. By Charles W. Girwin and E. B. Witmer, Deepwater Chemical Co., Ltd.

Operations of the Pacific Coast Borax Co. By G. A. Connell, technical director, Pacific Coast Borax Co.

ALL-DIVISIONS LUNCHEON 12:15 P.M.
TOWN HOUSE

George D. Dub, Chairman, Southern California Section, presiding.

TECHNICAL SESSION—2 P.M.
EL VENADO ROOM, ELKS CLUB

Stuart H. Ingram and M. Harrison Evans, Co-chairmen.

Tunnel Driving for Owens River Gorge Project. By J. D. Laughlin, assistant engineer of Design and Construction, Los Angeles Dept. of Water and Power.

Crushing by Impact. By W. W. West, chief sales engineer, Pennsylvania Crusher Co.

Bastnasite Discoveries Near Mountain Pass, California. By Lloyd C. Pray and Wm. N. Sharp, California Institute of Technology and U. S. Geological Survey.

ANNUAL BUSINESS MEETING OF THE
MINING BRANCH—4 P.M.

Ian Campbell, Chairman, Mining Branch, Southern California Section, presiding. Reports, nomination of officers, miscellaneous business.

EVENING ENTERTAINMENT

Dinner dance, given by the Petroleum Branch. All Institute members and their guests are cordially invited.

Call for Information on Use of Geophysics

Sherwin F. Kelly, of S. F. Kelly Geophysical Services, Inc., Rm. 318, 900 Market St., Wilmington, Del., has issued a call for information from geophysicists, geologists and company and government officials so that he may prepare a paper for presentation before next year's annual meeting in St. Louis.

Mr. Kelly says: "Geophysical techniques can properly be considered as only one step, or only one method to be employed by exploration geologists, engineers and executives in narrowing their search for ore. The geophysical data must furthermore be integrated with geological information and drill results. Any discovery of ore can therefore not be attributed directly to geophysics, but only to a program of exploration in which geophysics played a role. Therefore, I wish to compile data on those deposits whose discovery has been expedited or facilitated as a result of the inclusion of geophysical methods in the exploration program.

The material which I require can be grouped under the following categories.

"1. A list of orebodies with which you are acquainted and in whose discovery program geophysical techniques were included. What geophysical methods were employed in each case? 2. What was the approximate date of each discovery? 3. An estimate, or guess, as to what the geophysical survey cost in each case. 4. Information on the estimated total tonnage of ore and its grade, in each such deposit, or estimated total value of the ore; or,—5. Value of the production to date from those bodies which are, or have been actively exploited. 6. An estimate, or rough guess, as to the possible total expenditure for geophysical exploration for mineral deposits in the province, state, or country in which the above discoveries are located; or, a rough estimate of the probable total expenditure for geophysical exploration by your company. 7. Any information you can give me on expenditures for geophysical exploration in the mining industry, which will help me arrive at some sort of total figure, will be deeply appreciated. I am trying to arrive at an estimate of the total expenditures for geophysics in the mining industry since 1920."

James Draper Francis—Rand Medalist

Announcement was made at the Aug. 15 joint meeting of the Executive and Finance Committees that Howard N. Eavenson, Chairman of the Rand Foundation Award Committee, had recommended to the AIME Board of Directors that the Rand Medal for Mining Administration for 1950 be awarded to James Draper Francis of Huntington, W. Va., Chairman of the Board of Directors of Island Creek Coal Co. and Pond Creek Coal Co. and various affiliates, with the following citation.

"For successfully administering coal properties for more than thirty-five years until these two companies have become one of the largest and most successful units in the coal industry; for opening and managing new properties, for improvements in marketing and business methods, for his general interest in all industrial matters, his continued interest in research and his excellent citizenship in promoting not only his own, but all other interests in his general community."

The report was accepted, and the recommendation of the Committee was approved.

Coming Events

Oct. 3-5, American Institute of Electrical Engineers, Baltimore.

Oct. 5, AIME, Ajo Subsection, Ajo, Ariz.

Oct. 4-6, AIME, Petroleum Branch, Roosevelt Hotel, New Orleans.

Oct. 5-7, Canadian Institute of Mining & Metallurgy, district no. 4, annual convention, Red Lake, Ont.

Oct. 11-12, AIME, Industrial Minerals Div., Rocky Mt. Section, El Paso, Texas.

Oct. 12, AIME, El Paso Metals Section, El Paso, Texas.

Oct. 12-13, AIME, Southern California Section, Metal, Mining, and Petroleum Branches, Elks Club, Los Angeles.

Oct. 13, AIME, Southwestern Section, Open Hearth Committee, Iron and Steel Div., Houston, Texas.

Oct. 13-14, International Mining Days, El Paso, Texas.

Oct. 13-14, South Dakota Conference on Industrial Research & Development, annual conference, Alex Johnson Hotel, Rapid City, S. Dak.

Oct. 16-20, National Safety Congress and Exposition. Industrial safety sessions, Stevens, Congress, and Morrison Hotels; traffic safety, Congress; commercial vehicle, farm and home safety, La Salle; school, Morrison.

Oct. 17-20, AIME, Industrial Minerals Div., fall meeting, Norman, Okla.

Oct. 20, AIME, Eastern Section, Open Hearth Committee, Iron and Steel Div., fall meeting, Warwick Hotel, Philadelphia.

Oct. 20-21, Engineers' Council for Professional Development, annual meeting, Hotel Tudor Arms, Cleveland.

Oct. 22-24, American Mining Congress, metal and nonmetallic convention, Biltmore Hotel, Los Angeles.

Oct. 23-25, AIME, Coal Div., and **ASME,** Fuels Div., Hotel Statler, Cleveland.

Oct. 23-27 National Metal Congress and Exposition, International Amphitheater, Chicago. Participating organizations: AIME, Headquarters, Hotel Sheraton; ASM, Headquarters, Palmer House; American Welding Society, Headquarters, Hotel Sherman; Society for Non-destructive Testing.

Oct. 27, AIME, Southern Ohio Section, Open Hearth Committee, Iron and Steel Div., annual meeting, Deshler-Wallick Hotel, Columbus, Ohio.

Oct. 27, AIME, Lehigh Valley Section, Traylor Hotel, Allentown, Pa.

Nov. 3, AIME, Pittsburgh Section of Open Hearth Committee, Iron and Steel Div., and Pittsburgh Section, AIME, annual meeting, William Penn Hotel, Pittsburgh.

Nov. 3-5, New Mexico Geological Society, field conference, San Juan Basin.

Nov. 9, American Mining Congress, Coal Div. Conference, William Penn Hotel, Pittsburgh.

Nov. 14, AIME, Buffalo Section, Open Hearth Committee, Iron and Steel, Div., all-day meeting, Statler Hotel, Buffalo.

Nov. 15, AIME, Mining Branch, Southern California Section, joint meeting with AAPG.

Nov. 17, Illinois Mining Institute, annual coal meeting and banquet, Hotel Abraham Lincoln, Springfield, Ill.

Dec. 7-9, AIME, Electric Furnace Steel Conference, Iron and Steel Div., Hotel William Penn, Pittsburgh.

Feb. 18-22, AIME, annual meeting, Jefferson Hotel, St. Louis. Metals Branch session to be held at the Statler Hotel.

Drift of Things . . . . as followed by *Edward H. Robie*

At Salt Lake City

This is a warm Sunday morning in Salt Lake City at the end of a hectic convention week. Some 5400 mining men have stormed "The Crossroads of the West," a new record by far. Today the Sunday quiet is welcome, even though we had to eat breakfast alone. We have just returned from the Temple grounds where, from 8:30 to 9, we heard the 1098th Sunday broadcast of the Tabernacle choir and organ. Thousands of others were there too. Having heard it in their homes they were curious. They were not let down, for the setting is most impressive. After the formal program we listened to a 20-min rehearsal of a future program. Also, we introduced ourselves to Richard Evans, he of "the spoken word." Mr. Evans is a bespectacled, slender, black-haired, earnest, pleasant man of about 40. He was formerly a radio announcer on station KSL. He himself writes the little 3-min sermons that he delivers so well. This, he says, enables him to voice them much more sincerely and effectively. He has, perhaps, done more than anyone else to enkindle a kindly feeling towards Mormons in the minds of those who have few other contacts with the group.

When in Salt Lake one of the interesting things to do is to participate in one of the guided tours through the Temple grounds. This morning the groups were large. To what extent they were convinced of the authenticity of the story of the founding of the Mormon Church through the revelation bestowed on teen-age Joseph Smith at Palmyra, N. Y. more than a hundred years ago we do not know. The guide admitted the episodes sounded fantastic. No more so, however, than the early history of other religions. Though unconvinced, one would indeed be rash to say that miracles cannot happen. The modern, like the ancient world, exhibits too many of them.

Salt Lake is one of our favorite cities. It is clean and well kept. Its streets, some 90 ft wide, were laid out with surprising vision. Water continuously flows down the gutters of Main St. The busses are unobtrusive and quiet. The water has a good taste, with no suggestion of chlorine. The people look like good Americans. The only place, so far as we see, where liquor is sold is in a comparatively obscure store, not listed in the telephone book, run by the Utah Liquor Control Commission. We accompanied one of our renegade friends who knew his way there. One must have a permit, and purchase of more than a bottle or two is viewed with some suspicion. The 3.2 pct "beer" sold in some of the restaurants does not seem too popular.

The Hotel Utah, owned by the Mormon Church, is one of America's fine hotels, reasonably priced, well kept, staffed and provisioned. The young ladies who greet you at the desk, run the elevators, and wait on the tables are attractive and well groomed. Practically not a red fingernail in the lot, and is that a relief. The Starlite Gardens on the 10th floor is equally as attractive as the Starlite Roof of the Waldorf-Astoria in New York, and the absence of cocktails lends it a unique atmosphere. To be sure, one may be disillusioned if one peeks under the tables, but that isn't fair. In the Coffee Shop—and won't someone please think up a new name for an informal restaurant—one may have the privilege of breakfasting off a Cranshaw melon and a Utah Pioneer griddle cake. The paper money one gets in change is all brand new. Even the half dollars and quarters are polished to mint-brightness, a stunt that we believe the Hotel Davenport in Spokane started, and which the St. Francis Hotel in San Francisco practices. The Treasury Dept., as we recall, got after them for presumably illegally recovering the silver from the operation, but the hotel proved that it all went down the drain. If they wasted it, it was all right.

All of which is introductory to saying that the AIME has a new field office in Salt Lake. Roy E. O'Brien, an Anaconda man, has been appropriately embued with enthusiasm for the AIME, has passed the elementary examination on what makes it tick and why one should belong, and has set up an office in the Newhouse Bldg., 10 Exchange Place, as Western Secretary of the Mining Branch. He will be glad to see his many friends there and to do what he can for AIME members west of the Mississippi. Even metallurgists and oil men will be welcomed if they don't stay too long. Much of Roy's time, though, will be spent in the field, helping members in the more remote places who may have problems, assisting in building up Local Sections, promoting interchange of technical information, and interesting new people in the Institute. Any members wishing him to call on them when in their neighborhood should drop him a line. The success of Bill Strang and Joe Alford in establishing the Dallas office of the Petroleum Branch presages an equally bright future for the Salt Lake office.

We have already mentioned the reason why we are here—the 1950 Metal Mining Convention and Exposition of the Western Division of the American Mining Congress. Julian Conover and his staff did a marvelous job. Of the record attendance of some 5588, 4525 were operating men, suppliers, and 1063 ladies. The exhibits were worthy of attendance, the 125-odd exhibitors having their newest models on display in profusion. The AIME had a booth for the first time in recent years. The Institute's meetings do not feature machinery and equipment, except on a small scale by the Petroleum Branch, so the Mining Congress offers a unique opportunity for sales engineers and operating men to get together. Problems of economics, taxation, and legislation loomed large in the group meetings. We did not attend too many sessions, nor stay too long, the temperature being what it was. But we gathered that the small mine still has its ups and downs—mostly the latter so far this year; that the Government isn't doing what it should; and that there is a difference of opinion as to just what action we should take. We predict that much the same talk will be heard at the Los Angeles meeting of the Congress next year.

Utah's Governor, J. Bracken Lee, made a wonderful impression on everyone at the Welcoming Luncheon. He talks like the kind of man we need at Washington. As to whether his actions are in accord with his words, the comment of Utahns (we don't like that term much either) is favorable, but we are just naturally suspicious after the experience of the last two decades. How about Eisenhower and Lee in 1952?

Bingham Canyon was the destination on the first night. An al fresco spaghetti and bean supper was served on long tables on the new parking lot overlooking the pit. The "Galena Days" celebration was on, which means the natives had grown beards, the store windows were full of old-time signs, and gambling games were unrestricted. Bingham is the greatest man-made sight we have ever seen—maybe the greatest in the world, but it should be viewed from the far rim. About 175,000 tons is moved a day, some 75,000 tons of which is ore, containing around 0.8 pct copper.

AIME Personals

H. Travis Adams is now employed by the Trans-Continental & Western Airlines as staff assistant to the system fuel administrator.

Richard C. Anderson has joined the Fresnillo Co., Fresnillo, Zacatecas, Mexico.

Martin Armagnac has resigned his position with the San Francisco Mines of Mexico Ltd., San Francisco del Oro, Chihuahua, Mexico. He is now employed by the Fresnillo Co., Fresnillo, Zacatecas, Mexico.

August J. Breitenstein has been appointed to the Mining Advisory group of the Bituminous Coal Research, Inc. Mining Development committee. This is in connection with the coal industry's program to develop improved mining equipment and methods.

William F. Betzler, Sr. is now employed by the Interstate Iron Co., Virginia, Minn. as a mining engineer.

Oliver Bowles

Oliver Bowles, who retired from the U. S. Bureau of Mines in 1947, is again on the staff in the capacity of a consultant dealing with special phases of strategic minerals.

Vsevolod A. Gorsky has joined the Incorporated Gold Mines Ltd., N. Transvaal, as manager of the Knights Pietersburg Gold Mine.

M. W. Jasper is now a mining engineer and geologist for Western Uranium Cobalt Mines, Ltd., S. Hazelton, B. C.

S. G. Lasky of the U. S. Geological Survey is one of six civilians from the civilian agencies of the government who have been detailed to attend the Industrial College of the Armed Forces for the current academic year.

William W. Lynch, mining engineer, has been elected vice-president of the Calumet & Hecla Consolidated Copper Co. He has been New York representative of the firm.

William Maratta has accepted a position in the engineering department of Clinchfield Coal Corp. His previous connection was with Eavenson & Auchmuty.

Richard E. Mieritz has joined the Northern Peru Mining & Smelting Co., Tacna, Peru. He was formerly connected with the American Smelting & Refining Co., Tucson.

T. M. Patten is general superintendent for the Silas Mason Co., Grand Island, Nebr. He will be working on the modernization and operation of a one load line at Cornhusker Ordnance plant for loading rockets.

S. Lewis Rohrer

S. Lewis Rohrer recently took the examination given by the State Board of Registered Professional Engineers in Nevada and is now a professional mining engineer in that state. He will remain district materials engineer for the California State Div. of Highways, District IX, Bishop, Calif.

Earl L. H. Sackett, formerly assistant chief metallurgist, Baroid Sales Div., National Lead Co., Malvern, Ark., is now superintendent, Washington County, Missouri operations for the same company.

Stuart St. Clair left in August for an extended trip in the East Indies. He will be engaged in mineral and mine examination work and is expected to be absent from New York for approximately 3 months.

D. S. Sanders

D. S. Sanders has accepted the position of consulting metallurgist with the Sociedad Minera Puquio-Cocha, Morococha, Peru. He was previously superintendent of concentrators, Cerro de Pasco Copper Corp. He expects to return to the States before the end of November.

John H. Schissler, Jr. has recently resigned his position as contract and bonus engineer for the Copper Queen Branch of the Phelps Dodge Corp., Fresnillo, Mexico. He has accepted the position of chief engineer with the Fresnillo Co., Fresnillo, Zacatecas, Mexico.

Glenn H. Sides is now employed as an industrial engineer, coal mines, with the Tennessee Coal, Iron & Railroad Co., Birmingham, Ala.

H. De Witt Smith has left for French Equatorial Africa and South Africa. He will expect to return to New York about October 15.

J. N. Steingasser is now a rodman for the Eastern Gas & Fuel Associates, Wharton, W. Va.

Walter A. Sterling recently elected vice-president, Cleveland-Cliffs Iron Co. is now directing extensive mining operations of the company and will be located in Cleveland. He was previously manager of the company's Mesabi range. **Grover J. Holt** is succeeding Mr. Sterling and is now manager of the Mesabi range properties at Hibbing. **C. W. Allen** is general manager of the iron ore operations at Marquette and Masabi in Ishpeming. He is succeeding **C. J. Stakel,** who is retiring, but will remain as an acting consultant.

J. P. Warner has been working for

Sherritt Gordon Mines Ltd., at their pilot plant in Ottawa.

Harry J. Wolf

Harry J. Wolf recently has been engaged in consulting work relating to mining ventures in Colorado and mineral deposits in North Carolina.

Charles Will Wright recently spent a 10-day trip in Rio de Janeiro.

Richard Vaughn Wyman has joined the Cerro de Pasco Copper Corp., Cerro de Pasco, Peru as assistant geologist.

—Obituaries—

Harold S. Arnold (Member 1945), former assistant to the president, International Nickel Co., Inc., died suddenly on August 7. Mr. Arnold was born in North Abington, Mass. on Oct. 19, 1886. He attended the Phillips Exeter Academy from 1902 until 1906 and then entered M.I.T., graduating in 1910. He then joined U. S. Smelting, Refining & Mining Co. as a miner at Kennet, Calif. For the next few years he was engaged in general mining activities in California, Arizona, and Mexico. In 1914 he joined International Nickel Co.'s physical testing lab in Bayonne, N. J. When the operating and technical department at Bayonne was formed, Mr. Arnold was made assistant to the superintendent. He worked in the planning and construction of the company's Huntington works in Huntington, W. Va. and later became production manager of that plant, operating from the New York office. In 1935 he was appointed technical assistant to the vice-president and in 1949, technical assistant to the president. During the past 15 years he had acted as technical advisor on all motion pictures produced by the company.

John F. Berry (Member 1893), former mechanical engineer has died.

He spent most of his career in various mining positions in South Africa. In 1937 he came to Mexico and was employed as superintendent for the Cia. Minera Asarco, Aire Libre, Pue.

Otis Adams Critchett (Member 1927), died on July 11 after a long illness. Born in 1875 at Monroe, Mich., he attended school there, and in 1897 graduated from the University of Michigan with the degree of Ph.C. He was employed by the Candelaria Mining Co. for one year as a chemist, and in 1902 he went into the assay business at the Custom Assay Office. He then joined the firm of Ferguson & Critchett, the southwest's oldest assay office, as an assayer and chemist. He also did chemical and metallurgical consulting work.

M. G. Driessen (Member 1949), died on July 16 in Switzerland following an emergency appendectomy. He had attended the International Conference on Coal Preparation in Paris and was visiting in Switzerland when stricken. Born in Hengelo, Holland in 1899, Mr. Driessen was a graduate of the University of Zurich, Switzerland and the University of Delft, Holland. In 1923 he came to America and spent two years with the Westinghouse Electric Corp. and DeLaval Steam Turbine Co. He returned to Switzerland in 1925 to do research work for the Brown Bovéri Co., Baden. Mr. Driessen became chief of research for the State Mines of Holland in 1929. While there he invented a device for the concentration of heavy media in coal cleaning plants. In January 1948 he returned to the United States and joined Heyl & Patterson, Inc. He was an active member in the Coal Division of AIME and wrote many articles on coal preparation.

Hugh F. Marriott (Member 1905), died in 1949. He was born in London in 1869, and attended the Royal School of Mines from 1886 to 1891. He spent some time working on base metal and silver mines in the Spanish Pyrenees, and also in Johannesburg, South Africa, doing prospecting and geological work, and writing scientific reports on the earth temperatures, air supply to mines, etc. In 1930 he joined the Panama Corp. in London and in 1936 was president of the Panama Corp. (Canada).

Warden A. Moller (Member 1906), died on May 29. Born in Shanghai in 1871, he spent most of his life in the Far East. He was employed as a mining engineer in the mining department of several companies in Tientsin, Pei Piao, and Manchuria. In 1930 he went to England and in 1931 became associated with Hooper, Struve & Co., Ltd., London.

Edwin J. Prindle (Member 1914), former mechanical engineer, is dead. Born in 1868 in Washington, D. C., he received the degree of mechanical engineer at Lehigh University in 1890. He also attended the National University in Washington, D. C., and received his Bachelor of Laws and then a Master's degree. He was employed as an examiner for the U. S. Patent Office and later became engaged in the practice of patent law involving cases relating to mining, metallurgy, geology and chemistry. He was a member of the firm of Prindle, Wright, Neal & Bean.

Samuel Alfred Taylor

An Appreciation by

Newell G. Alford

Ending 62 years of devotion to his profession, the Institute's 47th President, Samuel Alfred Taylor, died on August 20th after a brief illness at his home in Pittsburgh, Pa. While his active interests in engineering and coal mining were broad in an international sense, he was a leader in the educational, religious, and civic development of his native community.

A western Pennsylvanian throughout his 86 years, Mr. Taylor was born in North Versailles Township, Allegheny County, on October 24, 1863. Educated locally, he graduated from Western University of Pennsylvania, now University of Pittsburgh, in 1887 with the degree of Civil Engineer and at the same university was awarded the degree of Doctor of Science in 1919. Beside being one of his Alma Mater's oldest graduates, Mr. Taylor served as vice-president of its Board of Trustees for many years. He acted as Dean of its School of Mines from 1910 to 1912, inclusive, while continuing his consulting practice.

Before establishing his consulting practice in Pittsburgh in 1905, Mr. Taylor served in many phases of structural, municipal, and mining engineering and was employed in the engineering departments of Carnegie Steel Co. and the Pennsylvania Railroad.

With seasoned experience and understanding in his field, his judgment became acknowledged by the district's contemporary masters in steel, coal, and banking. His reminiscences were studded with gems from his personal experiences with these men.

His professional work had taken Mr. Taylor to practically all of the coal fields in North America. He was one of the leading spirits in the formation of the National Coal Assn. in the decade before World War I. He was technical advisor to the U. S. Fuel Administration during World War I, along with Rembrandt Peale and John P. White. Later, he served as a member of the Engineering Valuation Committee on the U. S. Coal Commission.

Mr. Taylor joined the Institute in 1905. His term as President of the Institute in 1926 marked the substantial beginning of active interest by coal mining engineers in Institute affairs. He served as an Institute Director from 1915 to 1920, inclusive, and again in 1927 and 1928.

Mr. Taylor had been President of the Coal Mining Institute of America in 1911, of the American Mining Congress in 1912, and of the Engineers' Society of Western Pennsylvania in 1913.

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on July 31, 1950, was 16,466; in addition 4261 Student Associates were enrolled.

ADMISSION COMMITTEE

E. C. Meagher, Chairman; Albert J. Phillips, Vice-Chairman; George B. Corless, H. P. Croft, Lloyd C. Gibson, Ivan A. Given, F. W. Hanson, T. D. Jones, P. Malozemoff, Richard D. Mollison, and John Sherman.

Institute members are urged to review this list as soon as the issue is received and immediately to write the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

Alabama

Dixiana—Risser, Hubert E. (M) (R.C/S—J-M).

Arizona
Naco—Humphrey, William A. (J).
Phoenix—Brucker, Fredric Louis (M).
Ray—Zaskalicky, Michael Francis (J) (C/S—S-J.)

Arkansas
Bauxite—Erspamer, Ernest Gordon (J) (C/S—S-J).

California
Van Nuys—Slosson, James E. (J) (C/S—S-J).

Colorado
Climax—Rochin, Hector Anton (J) (C/S—S-J).
Denver—Shubart, Stanley C. (M).
Ouray—Bell, Franklin A. (M).

District of Columbia
Washington—Mulligan, Ralph C. (M).

Florida
Lakeland—Enright, Charles A. (M).

Illinois
Chicago—Golden, Richard M. (J).
Chicago—Raymond, John G. (J).
Chicago—Rinnander, Adolph J. (M).
Moline—Herbert, Charles F. (R.C/S—J-M).

Iowa
Council Bluffs—Gregory, Thomas L. (C/S—S-J).

Kansas
Pittsburg—Blair, John H. Jr. (J) (C/S—S-J).

Kentucky
Madisonville—Haley, Ted Davis (J) (C/S—S-J).

Michigan
Ishpeming—Fritts, Crawford E. (J) (C/S—S-J).
Ishpeming—Riedel, Robert W. (J) (C/S—S-J).

Montana
Butte—Brox, Gordon B. (J).

New Jersey
Trenton—Benoit, Maurice E. (J) (C/S—S-J).
West Point Pleasant—Schwab, Robert Charles (J) (C/S—S-J).

New Mexico
Albuquerque—Goldsmith, Louis H. (J) (C/S—S-J).
Carlsbad—Fuchsman, Charles H. (M) (C/S—J-M.)

New York
Long Island—Toman, Larry, Jr. (J) (C/S—S-J).
St. Lawrence County—Jones, Marshall G. (M)

North Dakota
Grand Forks—Fyten, John Herbert (J) (C/S—S-J)

Ohio
Cleveland—Litwinowicz, Vincent K. (C/S—S-J).
Columbus—Overstreet, William J. (A).
Green Springs—Muller, Max M.
Newark—Ratchen, James H. (J) (C/S—S-J).
Shaker Heights—Jackson, Herbert C. (A).

Pennsylvania
Harwick—Buzard, William W. (J) (C/S—S-J).
Lansford—Bolinger, John C., Jr. (A).
Lansford—Rickert, E. E. (M).
Lehighton—Shoenberger, Ronald W. (J) (C/S—S-J).
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